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J. W. RICHARDS, PH. D. President
E. F. ROEBER, PH. D. Editor

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Ten Years of Electrochemical Activity at Niagara Falls.

On Aug. 26, 1895, the Niagara Falls works of the Pittsburg Reduction Company started operation. On Oct. 19 of the same year the current was turned on at the plant of the Carborundum Co. Thus it is now just ten years that electrochemical activity started at Niagara, and in this short space of time more than a dozen varied electrochemical industries have grown up and are flourishing within a radius of 2 miles from the Falls. In the first two issues of our first volume Dr. J. W. Richards gave an extensive review of the electrochemical industries of Niagara, and later developments have been recorded from time to time in these columns, especially in a paper by Mr. F. A. J. FitzGerald in our last issue. It must, therefore, suffice to give here a brief summary of the principal products. In electric furnaces are made at Niagara artificial graphite, siloxicon, silicon, carborundum, alundum, calcium carbide, phosphorus and various ferro-alloys. By electrolysis of fused electrolytes are made aluminium, sodium (for the production of various important derivatives), and caustic soda and chlorine. The latter two products are also made by electrolysis of aqueous solutions; while other products of such processes are caustic potash and hydrochloric acid and chlorates. The only example of a process using electric discharges through gases on a commercial scale at Niagara is the production of ozone for the manufacture of vanillin. As the latest developments we may mention that ground has recently been broken for the caustic soda and chlorine works, which will use the Townsend diaphragm cell, while in the old barn in which Mr. Rossi made his pioneer experiments on ferrotitanium, Mr. Ruthenburg's process of agglomerating in the electric furnace magnetic iron concentrates, now undergoes an experimental trial. This is certainly a splendid development, and while as a necessary concomitant to the success achieved there have been commercial failures, yet their number is remarkably small. The most notable one is probably that of the Atmospheric Products Co., on the success of which very great hopes had been founded. When the company ceased operation, we understood from a reliable source that the process was in a "relatively" better shape than ever before, but presumably not in such a shape as to warrant the expenditure of further capital. In the interest of the problem it is greatly to be regretted that no exact information on the causes of the failure has been published. Certainly, the problem itself cannot be considered to be dead forever.

The foundation of the success of Niagara Falls as a center of electrochemical industries is the cheap power and the geographical location. The cost of power is far less than in any steam-generated plant; nevertheless, it is not low enough to

warrant, for instance, the manufacture of ferrosilicon at Niagara in competition with some European producers. The advantages of the geographical location of Niagara are manifold: Its location on the frontier of the United States and Canada, its easy accessibility by railways as well as by the waterway over the great Lakes, with their 3600 miles of shore line and navigable to the very docks at Niagara. Concerning the raw materials of the electrochemical industries of Niagara an interesting article by Mr. M. M. Green will be found in our present issue. But all these natural advantages and resources would not have been exploited without the ingenuity and fostering care of American electrochemists and engineers, who here proved that good engineers can be good business men. After all, in the wide sphere of industry it is the man who does things who counts more than he who knows things only. To push forward toward a fixed goal is specially promising in electrochemistry, and Mr. Acheson's single original experiment on the reaction between carbon and silica led, step by step, to the industries of carborundum, graphite, siloxicon, silicon. May the future of Niagara's electrochemical industries be as bright as the success of the first decennium.

Data for Metallurgical Calculations.

For his presidential address before the Society of Chemical Industry, Dr. William H. Nichols selected a subject which is of the most fundamental importance in modern industrial life, and which Dr. Nichols, as the foremost captain of chemical industry in the United States, is qualified to discuss as nobody else. This subject is the management of a chemical industrial organization, and an abstract of this address will be found on another page of this issue. While we hope to make some remarks on Dr. Nichols' address as a whole in our next issue, we wish to call attention in this note to one special point. This is the Statistical Department, which Dr. Nichols considers as absolutely necessary to intelligently run a large enterprise. This department is organized to compile and classify all the facts concerning the operation of the works; it advises the officers of the company each month of the cost of every product and step, and also of the profit or loss on each article and the total profit or loss of the company. These records represent an amount of information the value of which cannot be overestimated. With respect to this point it would seem that the metallurgical and chemical engineer has still very much to learn from the business man.

It is impossible to think of a business man who does not keep books, and who does not regularly make up his balance sheet of profit or loss. In metallurgical or chemical processes the energy balance sheet will necessarily always show a loss; for even if we had a perfectly ideal process, the principle of the conservation of energy requires that no more energy can be obtained from a process than has been put into it. Now, there will always be losses. But in this case it is just as important as for the business man to know exactly the efficiency of the process, not only of the whole process, but of each separate step and to locate the losses of energy. This is possible only by

making out a complete energy balance sheet. No business man would be satisfied with information to the effect that business is good or pretty good; he wants exact figures in dollars and cents. No metallurgical or chemical engineer should be satisfied before he can state exactly in figures what becomes of the calories or of the kilowatt-hours which he puts into his process. Nobody will say that this has been done in the past by engineers to anything like the same extent as is the rule of the business man. But this is not wholly the fault of the engineer.

The making out of a metallurgical or chemical energy balance sheet requires the knowledge of principles of physics and chemistry—these principles can be acquired, and we have been glad to learn from a good many of our readers how valuable and useful in this respect they consider the serial on metallurgical calculations by Dr. J. W. Richards, which is now being published in our columns. But to make out the energy balance sheet far more is required than mere principles of calculation. We need exact figures of specific heats, latent heats of fusion and volatilization, etc., and with respect to them, as a matter of fact, there is a woful lack of data. In many cases—important cases of chemical and metallurgical practice—the most fundamental figures are lacking which would be required for the calculation of the balance sheet. Interest in these matters has now been aroused, and this should go a far way towards getting the necessary data. But people should not expect to get those important and valuable data for nothing. It is usual to expect the best investigators and brightest experimenters to do this work for nothing, to pay the expenses of the work themselves, and then give the information to the world—all for glory. Undoubtedly investigators are willing to give their time and efforts, but they are usually unable to pay the expenses. They should, therefore, be assisted. The Carnegie Institution is now doing splendid work along these lines. So are many universities, by paying the expenses from special funds. So are many broadly-run industrial plants. But in the latter case there is still too much secrecy and not sufficient recognition of the fact that the self-interest of our manufacturers should animate them to give to the world, in the spirit of "do ut des," data of this kind obtained in their research laboratories. If every one will do his duty to the profession, then we will soon be able to collect the data which one would expect to get from a Statistical Department of chemical and metallurgical industries.

Sodium Sulphate Electrolysis.

The great success of electrochemistry in the industrial electrolysis of sodium chloride is well known and undisputed. Also well known is the commercial difficulty, due to the production of caustic soda and bleaching powder in equivalent proportions as long as all the chlorine is used for making bleaching powder, while the commercial demand for bleaching powder does not warrant this. Compared with sodium chloride electrolysis very little attention has been paid in the past to sodium sulphate electrolysis. The obvious reason is the difference in the available quantities of the raw materials. Nevertheless, there are sufficiently large deposits of sodium

sulphate which call for exploitation; and an indication of developments in this direction is an interesting patent granted to Dr. Kelly which is noticed in our Analysis of Current Electrochemical Patents in this issue. A comparison of sodium chloride and sulphate electrolysis suggests itself, although not much more can be attempted here than a discussion of the general aspects of the problem, especially as long as nothing of a definite nature is known concerning the purity of the sodium sulphate deposits; to prepare and maintain cheaply a pure electrolyte during operation is, of course, just as important in this case as in any other electrolytic process.

The cathodic reaction is the same in the electrolysis of sodium chloride and sodium sulphate; it is the production of caustic soda and evolution of hydrogen. As anodic reaction we have in the case of sodium chloride the evolution of chlorine gas which is to be worked up into further products. In the case of sulphate electrolysis, we have the production of sulphuric acid with the simultaneous evolution of oxygen gas. To compare the two cases we will assume that the same amount of caustic soda is to be produced. Then for the production of 2NaOH there are required in one case $1\text{Na}_2\text{SO}_4 + 3\text{H}_2\text{O}$, and in the other case $2\text{NaCl} + 2\text{H}_2\text{O}$ as starting materials. The products of the process are, besides the caustic, in the sulphate electrolysis $1\text{H}_2\text{SO}_4 + 2\text{H} + \text{O}$, and in the chloride electrolysis $2\text{Cl} + 2\text{H}$. In the latter case, by far most of the chlorine is still being worked up in practice into bleaching powder, while for the reason already mentioned there exists a distinct tendency to use the chlorine for better paying products. The only case of a commercial plant in which the anodic chlorine and the cathodic hydrogen are combined for the production of hydrochloric acid is that of the Roberts Chemical Co., of Niagara Falls, although in this case potassium chloride is the electrolyte. Electrolysis of sodium sulphate has the undisputable commercial advantage of yielding sulphuric acid for which there is absolutely no trouble to get a market. Besides, hydrogen and oxygen are produced in equivalent proportions; whether it would pay to compress, store and sell them for the uses of the oxy-hydrogen flame is a question which would depend mostly on the geographical position of the plant.

So far we have supposed that electrolysis of sodium sulphate offers no greater engineering difficulties than electrolysis of sodium chloride. The main point in both problems is, of course, to prevent the anodic and cathodic products from reaction with each other. Dr. Kelly, in his patent mentioned above, tries to solve this problem in a way which, as far as we know, has not been used in commercial sodium chloride electrolysis. He employs a three-compartment cell. The middle compartment is to contain the undecomposed solution of sodium sulphate, while the anolyte and catholyte are the resulting solutions of sulphuric acid and sodium hydroxide respectively. It is very doubtful whether diaphragms are alone sufficient without other special devices for removing the anodic and cathodic products from the electrolytic sphere of action. The reason is that these anodic and cathodic solutions formed by the action of the current will at once begin to act themselves

as electrolytes. There will, therefore, be a traveling of hydrogen ions from the anolyte into the middle department, and they will meet there SO_4 ions traveling in the opposite direction, thus forming sulphuric acid in the central part. On the other hand, from the catholyte OH ions will travel into the middle compartment, and will there meet sodium ions coming toward them, which results in the formation of sodium hydroxide. Thus both sulphuric acid and sodium hydroxide will be formed in the middle compartment. The circulation of the electrolyte will, of course, result in neutralization with the production of heat; this represents a loss of energy and an decreased efficiency of the process.

Moreover, the probability is small that equivalent quantities of sulphuric acid and sodium hydroxide are formed in the central compartment by the process just sketched. The proportion of the amounts of sulphuric acid and sodium hydroxide formed in the middle compartment depends essentially on the numerical values of the mobilities of the ions concerned. Without going into details we may say that calculation shows a slight tendency of sulphuric acid accumulating in the middle compartment, because a greater amount of sulphuric acid is formed in this department than sodium hydroxide. While any amount which is formed of the latter will be taken care of by being neutralized by an equivalent quantity of sulphuric acid, yet there will probably be left a balance of sulphuric acid in the central compartment, and this amount will increase during the operation. If this is so, the central compartment will get richer and richer in sulphuric acid and the main purpose of maintaining a neutral compartment will not have been accomplished. However, there is another action, that is, endosmosis, and almost nothing can be predicted *a priori* as to what it will do, especially since it depends on the nature of the diaphragms. This effect may be greater than that of ionic migration.

For sodium chloride electrolysis we have an abundant number of different means of removing the cathodic products of the current from the sphere of action. To mention only a few, in the Castner-Kellner process the sodium alloys with mercury, and is carried away by movement of the latter. In the Acker process, with its molten electrolyte, the sodium is absorbed by lead. Mr. Townsend's new diaphragm cell represents an ingenious method of carrying off by simple mechanical means the sodium hydroxide as soon as formed. Any of these or any other suitable means may, of course, also be applied at the cathode in sodium sulphate electrolysis. But at the anode we have a somewhat different problem. The anodic product, in case of an inert electrode, is sulphuric acid. This we may remove mechanically; or if we do not want its formation in the cell itself, we can catch the SO_4 ion by means of an anode forming an insoluble sulphate, like lead sulphate. In this case the sulphuric acid would be obtained afterwards. There can be no doubt that the problem can be solved if there is enough cheap and suitable raw material available to warrant the trouble of its solution. For the present it would be interesting to know what results have been obtained with Dr. Kelly's cell in practice.

Autumn Meeting American Electrochemical Society.

The eighth general meeting of the American Electrochemical Society is to be held at Bethlehem, Pa., on September 18, 19 and 20 (Monday, Tuesday and Wednesday), as was already announced in these columns.

Earlier than ever the preliminary programme has been issued, and while it not yet contains a list of the papers to be presented, yet in its general arrangement as to sessions, social functions, visits to plants and excursions, the programme is unusually elaborate and attractive.

While Monday is often considered a bad day for beginning a meeting, in the present case the programme has been so arranged that members in New York or Philadelphia may attend to their mail on Monday morning and leave New York on the Black Diamond Express at 11.55, or Philadelphia at 12.30, and arrive in time for the first session at 3.00 p. m.

Professional Sessions.—There will be three sessions for the reading and discussion of papers. On Monday at 3.00 p. m., on Tuesday at 9.00 a. m., and on Wednesday at 9.00 a. m. These three sessions will be held in the lecture room of the Physical Laboratory of Lehigh University, in South Bethlehem.

Visits to Plants and Excursions.

—These should prove extremely interesting and profitable to all members. On Tuesday afternoon the plant of the Lehigh Zinc Co., in South Bethlehem, will first be visited, which is one of the oldest and one of the most interesting zinc works in the United States. The Bethlehem Steel Co.'s world-renowned works will next be visited. For Wednesday afternoon an option of either of two excursions is offered. The first is by trolley to one of the big cement works in the vicinity, the other an excursion to Mauch Chunk, including a trip around the Switch-Back. Any who remain in Bethlehem Wednesday afternoon are cordially invited to attend the opening exercises of Lehigh University.

Social Functions.—A smoker will be held at one of the commodious club houses in Bethlehem on Monday evening. A lunch will be tendered to the visiting members by the authorities of Lehigh University, in the gymnasium, on Tuesday, September 19, at 12.30 p. m.

A subscription dinner will be served at the Eagle Hotel, Tuesday evening, September 19. The Eagle Hotel is to be the convention headquarters.

Entertainments for Ladies.—Besides the attractions which the beautifully situated historic town of Bethlehem offers in itself, a special programme of entertainment is provided for the visiting ladies. For Monday afternoon a carriage drive, starting from the Eagle Hotel at 3.00 p. m., has been arranged through the Bethlehems to South Mountain, with a visit to the grounds and buildings of Lehigh University. In the evening there will be an informal reception by the ladies' committee to the visiting ladies and their escorts at the residence of Professor and Mrs. J. W. Richards. On Tuesday morning an excursion will be made to Easton, with a visit to the large works of the Standard Silk Co.; lunch will be taken at Huntington, followed by a visit to Wygatt Mountain and grounds and buildings of Lafayette College, while in the evening a subscription dinner will follow, as already mentioned. For Wednesday morning an excursion is planned to Nazareth,

with visits to the quaint Moravian buildings, Nazareth Hall, school, Indian burying ground and the Whitefield house. For the afternoon the option of either an excursion to Mauch Chunk or the attendance of the opening exercises of Lehigh University is offered.

Dr. Joseph W. Richards, who has just sailed for Europe, is the chairman of the local committee; the secretary is Mr. Walter S. Landis, from whom any further information may be obtained.

The Management of a Chemical Industrial Organization

The official feature of the visit of American members of the Society of Chemical Industry to England was the annual meeting, held on July 10, in the University College, London, where Dr. Wm. H. Nichols delivered his presidential address, an abstract of which is given below. Concerning the trip through England and its social features and visits of plants—which we are informed is turning out the big success which had been expected—we will report when it is over, in our next

issue. The number of members is now 4326, compared with 4134 at the last annual meeting; these members live in sixty-four different countries, though the great divisions, the British Isles, United States, Australia, Canada, etc., are only reckoned as one each. The new president is Dr. E. Divers.

The subject of Dr. WILLIAM H. NICHOLS' extremely interesting and suggestive presidential address was "The Management of a Chemical Industrial Organization." Sir Henry Roscoe, later on, in proposing a vote of thanks to Dr. Nichols, characterized his address as "a lesson from America—a wonderful lesson, a most valuable lesson."

Dr. Nichols' plan of management is applicable in a large and rapidly growing country like the United States. The organization is divided into the following general

departments: Purchasing, Sales, Transportation, Finance, Construction, Operating, Research (or Investigation) and Statistical; and yet all of these must be so closely interwoven and work in such harmony that best results shall be obtained without delay. To attain this object, Dr. Nichols has found it necessary to arrange two committees, composed as follows: First, a manufacturing committee, consisting of the managers of the Operating, Construction, Purchasing and Investigation Departments, the chairman being the chairman of the executive committee of the board of directors. This brings all departments having to do with the turning out of products, present and prospective, in regular and systematic touch with each other and with the executive committee. Second, a sales committee, composed of the managers of the Sales, Operating and Purchasing Departments, together with a member of the executive committee. This also results in a close touch of the department distributing products with the others, and with the active governing body.

Concerning sales, Dr. Nichols expressed his firm belief that if the goods are well and economically produced—but not over-produced—at the right geographical points, their sale follows almost as a matter of course. "I do not pin my faith very strongly to that method of doing business, by means of which



WILLIAM H. NICHOLS.

manufacturers by agreement fix prices without much reference to cost, and thus court the competition which is sure to ensue. I believe that the best results in the long run are obtained by following natural laws willingly, rather than by combating them, invite the disaster which is certain to follow the temporary benefit so derived."

The main part of Dr. Nichols' address dealt with the organization of the Operating, Construction, Investigation and Statistic Departments.

OPERATING DEPARTMENT.

The Operating Department is one of great complexity and importance, involving, as it does, all the details of the management of the various works in connection with their corps of superintendents and chemists and the foremen, processmen and laborers under them. Its headquarters should be in the main office of the company, in close touch with the officers. A very complete and systematic organization and hearty coöperation with the other departments is necessary.

The administration of this department is carried on by a manager and assistant manager with a suitable clerical force. Through membership on committees and by frequent consultation with department heads and by correspondence with branch managers, the manager and assistant manager of the department are kept informed regarding the general policy and manufacturing needs of the company, and regulate manufacturing so far as possible in such manner as to provide a sufficiency of products wherever required.

This department naturally requires the services of a number of chemists, used for the daily routine of the work. The selection of these men is highly important, as from their number may come those who will go into the research laboratory of the company, or possibly fill its superintendents' positions, and in later years its offices. It is not sufficient that they have received a good education and have completed a technical course, but they must evidence such personal characteristics as lead to the belief that they are capable of advancement to positions requiring the exercise of thought and judgment, and the assumption of responsibility. It is desirable that they should have some knowledge of mechanical engineering and the general principles of construction.

The apprentice confines his duties for a number of months, and frequently for years, to a works laboratory, and incidental to his analytical work he gains a certain knowledge of the general routine which obtains at that plant. If he here displays ability, he is promoted to a position which will bring him into direct contact with the manufacturing process, and his duties will gradually change from those of analyst to those of a manufacturing assistant. Further promotions lead to assistant works superintendent and superintendent. The chemist must have a clear, logical mind, a singleness of purpose, and he must be able to separate the essential from the non-essential. This is very important.

CONSTRUCTION DEPARTMENT.

The Construction Department has entire charge of the designing and construction of all new plants, or parts of plants, and must act in the closest touch with the Investigation Department. The course of procedure was sketched in Dr. Nichols' address.

INVESTIGATION DEPARTMENT.

The Investigation Department deals with all the new propositions of a technical nature. A new proposition is in its control until sufficient data has been obtained to enable the Construction Department to design the necessary plant, if one be authorized by the executive committee. The organization of the Investigation Department should be sufficiently broad to permit the consideration of a manufacturing proposition from the points of view of the business man, the chemist, the engineer and the patent attorney. It consists of the manager, a chemical council, composed of the manager, the chief chemical

engineer, the chief chemist, who is director of the Research Laboratory, and such consulting chemists and engineers as the company employs. A corps of chemists on research laboratory work, an abstractor of current chemical literature, patent experts and a small office force complete the department staff.

The work of the Investigation Department originates from sources which may be divided into three classes: (a) The probability of reducing manufacturing costs; (b) a decision to produce well established products not previously manufactured by the company; (c) new applications of science to industry.

The commercial side of a new proposition calls for consideration of the following points: Its relation to the interest of the company, the market, manufacturing costs, investment necessary, source of raw materials, transportation.

On the technical side a study must be made of the process, other processes, raw materials, quality of product required.

The Research Department would not be complete without a laboratory plant, large enough to work out processes on a small manufacturing scale.

STATISTICAL DEPARTMENT.

The Statistical Department, which is absolutely essential in a company operating a number of plants, has to do with the compilation of facts and the deductions from them. It advises the officers within a reasonable time after the end of each month of the cost of every product and step, and also of the profit and loss on each article and the total profit or loss of the company. Dr. Nichols stated that the results compiled by the Statistical Department of his own company have been so exact that for several years the profits determined by public accountants at the end of the year have not varied 1 per cent from those which had been worked up in the Statistical Department month by month.

Copper Refining at the Kedabeg Copper Mines.

A long paper, read by Mr. Gustav Köller, before the Institution of Mining and Metallurgy in London, on March 16, 1905, gives an interesting description of this Russian electrified refinery.

The installation for the electrolytic process is situated in Kalakent. It was established in 1889 on a small scale and comprised originally only 36 electric baths, from which an output of 216,000 pounds of electrolytic copper was obtained annually from 90 per cent. raw copper. The slimes deposited at the anodes were low in precious metal, the maximum tenour never exceeding 2 per cent. of argentiferous gold. The process using raw copper was subsequently abandoned, and refined copper was substituted, the capacity of the works being also increased in 1893 to an annual production of 900,000 to 1,080,000 pounds of pure copper.

The plant now consists of 102 electric baths, which are supplied with current from a shunt-wound dynamo generating 700 amperes at 35 volts. These baths are grouped in four rows, and the arrangement in general differs from that usually adopted in other places only in that the baths are entirely independent of each other with regard to the circulation of the liquors. The circulation is effected by means of Siemens' geyser-pumps, two pumps being attached to each bath. This arrangement enables the baths to be worked either independently or in groups, and facilitates the control of each separate bath. They are made of pine wood, the length inside being 2 m., the depth 1.15 m. and the width 0.96 m. The joints are rendered tight with canvas lining, soaked in asphalt, a method which cannot be highly recommended, but which was adopted on account of the enormous price of lead in Russia.

In preparing the canvas lining, there are several precautions to observe, especially with regard to the quality of the asphalt or creosote. If the latter is insufficiently boiled before being applied, gases consisting of oxygen and carbon dioxide are

generated by the action of the free acid in the bath, and the cathode copper becomes oxidized by these.

The conductors which convey the current along the bath in the direction of its length have a cross section of 500 square mm., and the main conductors are slightly larger. The baths are charged with 15 anode and 14 cathode plates, beginning and closing with anodes. The electrodes in each bath are, of course, fixed in parallel, while the baths themselves, as whole units, are connected in series. The distance between the electrodes is 5 to 6 cc. The effective area of the anodes is 19 to 20 square m., and that of the cathodes 19 square m.

The anodes are of cast refined copper, containing 0.07 to 0.09 per cent. of precious metal, of which the 1-14th part is gold. The weight is 288 to 360 pounds, and they measure 90 by 86 cm., with a thickness of 2 to 2½ cm. The cathode sheets are 2 mm. thick, and before using are heated to redness to get rid of grease, and are provided with suspension hooks and metal strips to serve as terminals.

As already stated, two geyser-pumps are connected to each bath for circulating the electrolyte, an operation which must be carried on vigorously to ensure uniform deposition of the copper. The liquor is raised by atmospheric pressure, equal to about 70 mm. in the mercury column, through the delivery pipe from the neighborhood of the bottom of the bath and flows laterally into a distributing channel to mix again with the electrolyte at the surface. In this manner 800 litres of the liquor are circulated per hour in each of the baths.

The current density does not exceed 25 or 30 amperes per square m. of the cathode surface, the electromotive force across the baths being 0.18 to 0.25 volt. The current efficiency has been found to amount to 99 per cent.

The electrolyte is maintained at 75 grm. free H_2SO_4 per litre and the copper at 35 grm. per litre.

The purification of the electrolyte proceeds along orthodox lines by saturating with copper from calcined matte filtration through lean ore, evaporating with excess of anode scrap aided by air blows. The excess of copper brought into the liquor is crystallized out after evaporation.

The finished electrolytic copper has a degree of purity of at least 99.9 per cent., the average of the analyses giving 99.93 per cent. The weight of the finished plates is about 300 pounds.

The slimes are collected from the baths towards the end of the year, and by decanting with water are freed from the residual liquor, and are then screened and washed. In this condition they contain 35 to 40 per cent. of copper, of which amount 25 per cent. to 30 per cent. is extracted by heating 50° or 55° C., and treating with exhausted acid liquor. After washing again and drying they still retain 10 to 15 per cent. of copper, and from 25 to 30 per cent. of precious metal, and in this form they are sold, without further preparation, to the North German Refinery at Hamburg. In the year 1899 there were produced 863,136 pounds of electrolytic copper, and 306,612 pounds of anode slimes containing 722 pounds of silver and 55 pounds of gold.

CORRESPONDENCE.

Electric Furnace Experiments.

To the Editor of Electrochemical and Metallurgical Industry:

SIR: I beg to call attention to one point in Mr. Francis A. J. FitzGerald's very interesting article on "Materials for Resistors," published in the December number of 1904 of your journal. In referring to the curve in Fig. 4, on page 493, he says: "Since the voltage at the terminals of the resistor was constant, the curve represents the changes in resistance of the resistor as a whole."

But the figures that he gives in the table on the same page

show that the voltage at the terminals of the resistor was not constant, unless the resistance of the two carbon terminals was at least .2 ohm, which could not possibly have been the case with a furnace as shown in the diagram in Fig. 3 on page 492. According to the diagram the resistance of T1, T2 and M would be negligible, and the resistance of the resistor as a whole would be the sum of the resistance of G1 and G2. Now, the table on page 493 shows that the sum of the two voltages at G1 and G2 was not a constant, but in one case was 72 and in another as high as 84 volts.

This, of course, affects the shape of the resistance curve in Fig. 4, and an examination of the figures will show that the effect of making this correction would be to bring the points E and G lower down and B higher up; in other words, to simply accentuate the difference in the resistance caused by the weights, making the maximum total change in resistance a change of 55 per cent instead of 50 per cent, as the curve in Fig. 4 indicates.

JAMES J. DOYLE.

Chicago, Ill.

To the Editor of Electrochemical and Metallurgical Industry:

SIR: Mr. Doyle, in making his criticism, has overlooked the fact that the volts given in the table under G1 and G2 were not measured between T1 and M and T2 respectively, but between the rods A and B and the rods C and D. Therefore, the sum of the volts given under G1 and G2 at any time is the voltage between T1 and T2.

Although Mr. Doyle is mistaken in his criticism, there is an error in Fig. 4, for the Y-axis ought to be designated "reciprocals of current," or "resistance," instead of "ohms," and the upper figure marked on the axis should be 0.015, while the figure at the origin should be simply 0.

Niagara Falls, N. Y.

F. A. J. FITZGERALD.

Crystalline Structure of Electrolytic Copper.

To the Editor of Electrochemical and Metallurgical Industry:

SIR: The letter of Mr. Addicks, published in your issue of July, with photograph of crystalline electrolytic copper, recalls to my memory some experience I had many years ago, in 1883, while precipitating impure copper solutions obtained from the treatment of ores, when crystals similar to those described by your correspondent were obtained in the iron plates used for cementation.

The cold copper solutions were treated in vats filled with iron plates, chiefly cast iron; in one of the vats all the cement copper was deposited in crystals, instead of the usual form of cascara, and exactly in a fern-like grouping. The following observations were made as to the precipitation.

The copper deposited was more abundant in the edges than in the faces of the iron plates.

When two iron plates were placed in the copper solution facing each other, the copper deposit was more regular and adherent in the large plate, but more abundant, and forming crystalline groupings in the small plate.

In two iron plates placed, one showing its edge to the other face, the deposit was more abundant in the plate placed edge-wise. The deposit was crystalline and the larger part of it toward the edges.

In some of the iron plates the deposit was very irregular; the copper being deposited in crystalline parallel lines.

Some of the plates from old scrap had holes drilled in; the crystals were chiefly deposited in the edges of the holes, sometimes in the shape of spherical groupings.

When the solutions contained a considerable quantity of iron salts, most of the deposit was in crystals, some very large.

The high density of the current originated by the action of the solution on the elements forming the cast-iron plates, may have caused this crystalline deposition. This opinion was sustained by Bucholz in his classical experiment many years ago.

New York City.

J. BAXERES DE ALZUGARAY.

Heat Insulating Qualities of Refractory Materials.

The eminent importance of efficient heat insulation for the design of furnaces has been the subject of various discussions in our journal. Mr. F. A. J. Fitzgerald has shown repeatedly how great a saving of electrical energy may be accomplished by paying due attention to this point in the design of electric furnaces (our Vol. II., pp. 345, 439, etc.). A discussion of various refractory materials by Mr. E. K. Scott may be found in our Vol. III., p. 140. Mr. W. McA. Johnson recently (Vol. III., p. 214) discussed some principles of the use of refractory materials in general metallurgy, and pointed out how little exact knowledge we have concerning the heat-insulating quality of our ordinary refractories. A remarkable indication of the backward state of our knowledge of refractories is the fact that we use fire-clay generally as material for heat insulation as well as for transferring the heat from the fuel to the charge. This is, of course, really nonsensical. What is most thoroughly needed is exact data on the heat-insulating qualities of refractory materials.

With respect to this problem, a recent Faraday Society paper, by Messrs. R. S. HUTTON and J. R. BEARD, contains much useful information. The authors point out that the measurement of thermal conductivities of refractories is by no means difficult to carry out, at any rate to within an accuracy of 2 or 3 per cent, by means of a method and apparatus devised by C. H. LEES and J. D. CHORLTON (*Philosophical Magazine*, June, 1896 [5], Vol. XLI., p. 495 to 503). Messrs. Hutton and Beard have used this apparatus in their tests, carried out in the Physical Laboratories of Manchester University.

All the substances dealt with were either in the state of well-defined granular powders, or, in some few cases, where bricks were examined, these were previously broken up and finely powdered. The apparatus, as shown in Fig. 1, consists essentially of two thick brass discs, A and B, between which the material whose thermal conductivity is to be measured. The upper disc B is surmounted with a steam jacket S which is covered with felt, to diminish the loss of

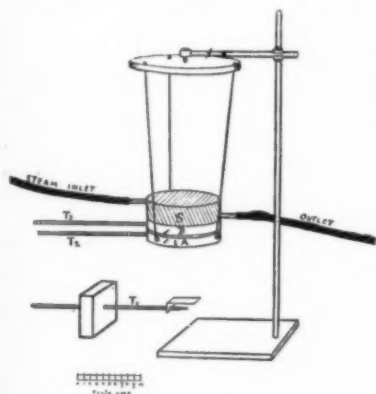


FIG. 1.—MEASUREMENT OF HEAT INSULATION.

heat. The lower disc A is allowed to cool by free conduction and radiation. This disc is suspended from a horizontal support by three strings. Both discs have the same dimensions, the diameter being 11.4 cm. and the thickness 1.3 cm. The powders are kept in position by a thin ring of red fiber, about 0.36 cm. high. A and B are provided with projecting pegs, the distance apart of which gives a measure of the thickness of the insulating material.

Three thermometers are used. T_1 registers the temperature of the air; it is provided with a screen to protect it from radiated heat from the lower disc. T_2 and T_3 indicate the temperature of the lower and upper disc respectively, and are inserted into radial holes drilled in the brass. When steam is passed through the steam jacket the temperature of the upper disc is raised to nearly 100°C ., heat flows through the material experimented on, and thus raises the temperature of the lower disc above that of the surrounding air. The lower disc loses heat by conduction and radiation to the air. Eventually

a steady state is reached when this loss of heat is equal to the heat received through the material experimented on.

The following equation enables the thermal conductivity K to be calculated from the result of such an experiment, provided h_1 , the external conductivity or emissivity, has been previously determined (from the rate of cooling of the lower disc at different temperatures):

$$K = \frac{t_2 - t_1}{t_3 - t_2} b \left(h_1 + h'_1 + \frac{phb}{2q} \right)$$

Here t_1 t_2 t_3 are the temperatures indicated by the thermometers T_1 T_2 T_3 respectively, and b denotes the thickness of the layer of powder in cm.; p is the perimeter and q the area of cross-section of the plate. Under the conditions of the experiment (*i. e.*, with the dimensions of the apparatus men-

tioned above) $h'_1 = 0.026 \sqrt{\frac{phk_1}{q}}$, where k_1 is the internal

conductivity of the metal disc (0.25 for brass). The conductivity k is the quantity of heat in gram-calories, which is transmitted per second through a plate 1 cm. thick per square centimeter of its surface, when the difference of temperature between the two faces of the plate is 1°C . According to the arrangement of the experiments, the conductivities refer to the temperature range from 20 to 100°C .

Messrs. Hutton and Beard found by this method the following conductivities, most of the granular powders tested just passed through a sieve with 600 meshes per sq. cm.:

Sand, white Calais.....	$K = 0.00060$
Carborundum, fine	0.00050
Carborundum, coarse	0.00051
Quartz "enamel"	0.00036
Quartz, fused	0.00039
Firebrick	0.00028
Retort graphite	0.00040
Lime	0.00029
Magnesia, fused	0.00047
" "Mabor" brick	0.00050
" "calcined Greek	0.00045
" "calcined "Veitsch"	0.00034
" "Pattinson's light calcined....	0.00016
Kieselguhr (infusorial earth).....	0.00013

RELATIVE VALUE OF DIFFERENT HEAT INSULATING MATERIALS AT HIGH TEMPERATURES.

Messrs. Hutton and Beard point out that it is not advisable to choose a material suitable for furnace uses from the above data alone.

Firstly, it is always necessary to know the general effect upon the substance under investigation of such temperatures as are likely to be experienced. Several, even amongst those materials mentioned in the above table, are unsuitable for sub-

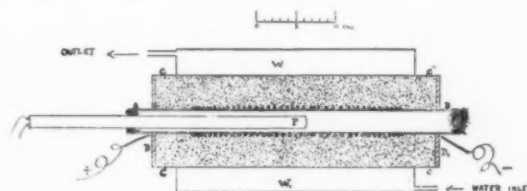


FIG. 2.—ELECTRIC FURNACE FOR TESTING REFRACTORIES.

jecting to high temperature on account of the physical changes, such as shrinkage, which they undergo. Chemical action, especially oxidation, also renders many insulators unsuitable for a large number of purposes.

Secondly, having by such considerations excluded all unsuitable materials from the choice, it becomes necessary to

compare the relative efficiency of those which remain under conditions more nearly approaching those experienced in practice. Messrs. Hutton and Beard have made the following tests:

An electric tube furnace, as shown in Fig. 2, was employed. A B is an unglazed porcelain tube with a heating coil of nickel wire wound on its central portion. The tube is kept in position at the center of the cylinder of sheet iron CCCC by washers of asbestos card DD. The cylinder C is surrounded by the water jacket WW. The temperature in the tube is indicated by the thermoelectric pyrometer P. The granular resistance material under test fills the space between the porcelain tube and sheet-iron cylinder.

The electric energy expended in the heating wire of nickel is kept constant over a considerable period by careful adjustment of an external resistance and constant observation of a precision wattmeter. The temperature at the center of the porcelain tube is noted at regular intervals, and from the observations a curve is plotted to show the relation between the rise of temperature and the time during which the heating has been in progress. The temperature of the water flowing through the external jacket was kept fairly constant; generally it varied from 2° C.

In such an experiment it would require a very long time for the temperature to attain a constant value. It was, therefore, found preferable, after the rate of increase in temperature had become rather slow, to reduce the power by a small but definite amount. Under these conditions the temperature falls, at first very rapidly, but soon reaches a constant value, or at any rate, falls so slowly that the variation is almost negligible. It is from a comparison of the heating curves with different materials and the same expenditure of power, and particularly

their efficiency to a considerable extent. Both substances exhibit large shrinkage at high temperature, and their chief uses are likely to be limited to external jacketing of furnaces lined with some more permanent but less good insulation.

Tests were then made to see what influence the preliminary heating might have on the results. Two similar experiments were carried out with the various substances. In the first case the preliminary stage of the heating was effected with 300 watts, in the second with 250 watts. In both cases the power was subsequently reduced to 200 watts, and then to 150 watts. The results showed that in all those cases in which no permanent alteration is brought about in the materials under test, the horizontal portions of the curve are practically identical, whatever the preliminary treatment had been. In earlier experiments, in which no water cooling had been used to maintain constant the temperature of the outer jacket of the enclosure, some anomalous results were obtained.

In their conclusions Messrs. Hutton and Beard point out that while bricks and general jacketing materials for furnaces should be of low thermal conductivity, it is of no less importance to choose crucibles, retorts and other containing vessels which are heated externally of as high conductivity as is feasible in conjunction with their other necessary qualities.

By a judicious use of the different materials and a stratified form of construction, it should be possible usefully to employ such excellent insulators as infusorial earth without risk of the damage which is inevitable if they be permanently subjected to too high a temperature.

On the Fundamental Formulæ of Chemical and Electrical Energy.

BY PROF. F. HABER, PH. D.

In the development of our knowledge of the relation between heat of reaction and energy of reaction, three different stages may be clearly distinguished. The oldest view is that which assumed the energy of reaction to be equal to the reaction heat. This view has been defended by Berthelot under the name of Principe du travail maximal for a long time. Sir William Thomson (Lord Kelvin)¹ has applied it to galvanic cells by connecting the reaction heat Q of the reaction which generates electric current, with e.m.f. E of the galvanic cell and with the valence n of the material involved in the reaction by means of the equations

$$(1) A = Q$$

and

$$A = 23,110 En.$$

Where A represents the energy of the reaction, and 23,110 follows from Faraday's electrochemical equivalent and from the electrical equivalent of heat.

The second stage of the development was inaugurated by Helmholtz, who derived from the principles of thermodynamics the result that A and Q differ from each other by variable quantity according to the equation

$$(2) A = Q + T \frac{dA}{dT}$$

or

$$(3) E = \frac{Q}{23,110 \times n} + T \frac{dE}{dT}$$

where T represents the absolute temperature, while the quantity $\frac{dA}{dT}$ or $\frac{dE}{dT}$ is called the temperature coefficient. The

term $T \frac{dA}{dT}$ represents that portion of the energy of reaction which is produced at the temperature T by the change of

¹Helmholtz' first opinion was the same.

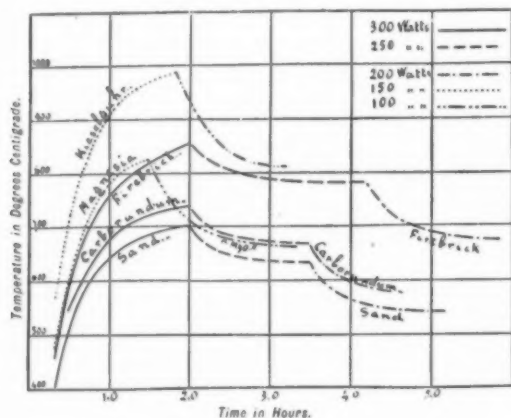


FIG. 3.—TESTS OF VARIOUS REFRACTORIES.

from the nearly horizontal portion of the curves with lower power expenditure that valuable conclusions can be drawn.

Fig. 3 illustrates some of the results obtained by Messrs. Hutton and Beard:

In the case of firebrick, carborundum and sand, the preliminary heating was effected with 300 watts, and after the lapse of about two hours the power was reduced to 250 watts, being kept constant at this for a period of one and a half to two hours, by which time the temperature had become almost constant. The curves for kieselguhr or infusorial earth (see FitzGerald, our Vol. III., p. 55) and light magnesia clearly show the very much smaller conductivity of these materials. With these two substances the preliminary heating was effected with an expenditure of only 150 watts, but despite this the rise in temperature is more rapid than with any of the other materials mentioned.

Unfortunately, neither light magnesia nor infusorial earth can withstand any high temperatures for long without losing

a certain quantity of external heat q into work. By considering an experimental arrangement, which has also been realized in practice, it is easy to make these relations clear.

We place a galvanic cell, together with its circuit, into an ice-calorimeter and measure the heat change which takes place when an electric quantity of 96,540 n coulombs passes through the circuit. The heat developed is equal to Q , that is, equal to the reaction heat of the chemical change taking place in the cell. The production of electrical energy is in this case only an intermediate process, and the resultant heat is the same as though the same reaction had taken place, to the same extent, in the ordinary chemical manner, by direct transformation of chemical energy into heat at zero degree C. without any generation of electric current.

We now modify this experiment by using two ice-calorimeters instead of one. One calorimeter contains the cell, the other one the external circuit, the resistance of which is made extremely large compared with the internal resistance of the cell; 96,540 n coulombs may again pass through the circuit. Now, several different cases are possible. The first case is that in the second calorimeter the heat Q is set free, while the first calorimeter, which contains the cell, shows no heat change. Then we have again $A = Q$. But this is not necessarily so, since the heat set free in the second calorimeter may differ from Q by a positive or negative quantity q . If in the second calorimeter the heat $Q + q$ is set free, the law of conservation of energy requires that the same amount of heat q disappears in the first calorimeter. The galvanic cell, which is placed in this calorimeter, will then have the property of changing at constant temperature the heat q into electrical energy. In this case the electrical energy A , which is changed into heat in the second calorimeter, is

$$(4) A = Q + q$$

The former formula (2) contains, instead of the calorimetrically measurable heat q , the term

$$T \frac{dA}{dT} \text{ or } T \frac{dE}{dT} \text{ in (3)}$$

The identity of q and $T \frac{dA}{dT}$ follows from the Carnot-Clapeyron principle as to the work which can be obtained from heat by reversible cyclic processes, generally called the second principle of the mechanical theory of heat.

In order to apply the formula of Helmholtz to experimental investigations, the differential quotient $\frac{dA}{dT}$ is replaced by the

quotient of the differences $\frac{\Delta A}{\Delta T}$ which is obtained if A is determined at two temperatures, near together. This, for the reactions which produce electrical energy in galvanic cells, may be done in a very simple way, since, in this case, it is only necessary to measure the e.m.f. of the reversible cell at different temperatures which are near together. In this way it has been possible to confirm Helmholtz' formula in many cases.

This formula does not state, however, anything concerning the influence of a change of concentrations; but it is easy to show experimentally that any change in the concentrations of the substances used causes a change of the temperature coefficient. A deeper insight into the relation between free energy, or e.m.f. and the temperature, was then obtained by van't Hoff, who developed the formula

$$(5) 23,110 E n = R T \left\{ n K - \ln \frac{C_{for'} C_{for''} C_{for'''} \dots}{C_{dis'} C_{dis''} C_{dis'''} \dots} \right\}$$

Here C is the concentration, for' , for'' , for''' represent the substances formed during the reaction, dis' , dis'' ,

dis''' represent the substances consumed or disappearing during the reaction, v' , v'' , v''' and n' , n'' , n''' the corresponding numbers of molecules. This formula of van't Hoff is valid only as long as the materials follow the gas law, i. e. for solutions only as long as they are not too concentrated. The numerous investigations which were undertaken to confirm by experiment the formulæ (5) and (2), I would consider as the second stage in the development, while the first stage is characterized by the faith in the validity of Thomson's rule (equation 1). Placing the equations 3 and 5 side by side—

$$23,110 E n = Q + 23,110 n T \frac{dE}{dT}$$

$$23,110 E n = RT \ln K - RT \ln \frac{C_{for'} C_{for''} C_{for'''} \dots}{C_{dis'} C_{dis''} C_{dis'''} \dots} \quad (5a)$$

the question is suggested: may Q and $RT \ln K$ be interchanged? Experience shows that this is only possible in some cases. In other cases results are erroneous. The third stage of development is characterized by the attempts to elucidate by theory and experiment the relations between Q and $RT \ln K$.

From Helmholtz' fundamental paper, it may already be seen that the difference in the specific heat of the compounds decomposed and formed rules the relation sought. Lewis and Richards in the United States, and van't Hoff in Germany, have recently brought this half-forgotten view to the front, and inaugurated the third stage of development, to the theory of which this paper may perhaps add some slight simplification.

The equation (2) of Helmholtz is a differential equation, and can be integrated. For this purpose we may write it as follows:

$$-\frac{A}{T^2} + \frac{1}{T} \frac{dA}{dT} = -\frac{Q}{T^2}$$

The left hand is now a complete differential, and we may write the whole equation at once as follows:

$$(6) \frac{d\left(\frac{A}{T}\right)}{dT} = -\frac{Q}{T^2}$$

The equation (6) is exactly identical with the formula of Helmholtz, the difference being only in the manner in which the equation is written. By integrating equation (6) we get

$$(7) A = a T - T \int \frac{Q}{T^2} dT$$

In equation (2) of Helmholtz the concentrations are assumed to remain constant while the temperature changes. In the integrated equation (7) the term a , therefore, depends on the concentrations. In order to calculate the integral it is now necessary to express Q as function of T . For this purpose we write

$$(8) Q = Q_0 + \sigma T$$

where Q_0 is the reaction heat at the zero point of the absolute temperature scale, while σ represents the difference of the mean specific heats of the consumed and formed substances at constant volume. Here we assume with van't Hoff that it is sufficiently approximate to consider σ as constant with varying temperature. The calculation can easily be made more exact by putting

$$\sigma = \sigma' + \sigma'' T + \sigma''' T^2 + \dots$$

By making the above approximate assumption, and introducing the term of Q from (8) into 7, we get

$$(9) A = a T + Q_0 - \sigma T \ln T$$

Now, going back to 5a such concentrations may be chosen that the second term on the right hand vanishes, then

$$(10) A = 23,110 E n = RT \ln K = a' T + Q_0 - \sigma T \ln T$$

Here a' is the special value of a corresponding to the concen-

trations chosen. For any concentrations the equation becomes

$$(11) A = 23,110 E n = Q_0 - \sigma T \ln T - RT \ln \frac{C_{a'}^{for} C_{b'}^{for} C_{c'}^{for} \dots}{C_{a'}^{dis} C_{b'}^{dis} C_{c'}^{dis} \dots} + a' T$$

Now, Th. W. Richards and van't Hoff agree in the view that a' is zero, at least when the number of molecules formed and decomposed is the same as *e. g.* in



and in similar cases. The equation then becomes

$$(12) R \ln K = Q_0 - \sigma T \ln T$$

The equation (12) is remarkable, since it shows the possibility of calculating the equilibrium constant K of the reaction from the reaction heat, the temperature and the difference of the specific heats of the substances formed and decomposed during the reaction. If σ is zero, the reaction heat is independent of the temperature and equal to $RT \ln K$. The difference between the reaction heat and $RT \ln K$ on one hand and the change in the reaction heat by the temperature on the other hand, control each other, so that, by determining one, the other may be calculated.

On the assumption of Richards and van't Hoff that a' is zero, we get also from 11—

$$(13) \frac{dE}{dT} = -\sigma (1 + \ln T) - R \ln \frac{C_{a'}^{for} C_{b'}^{for} C_{c'}^{for} \dots}{C_{a'}^{dis} C_{b'}^{dis} C_{c'}^{dis} \dots}$$

By help of 13 the numerous investigations of dE/dT may be used to calculate σ when the concentrations are known too.

Unfortunately, there are very few determinations of the specific heat of dilute solutions. It appears very important to systematically study this matter, and we may hope that a general endeavor to this effect will prove not less fruitful than the work done in the investigation of the effect of concentration.

The equation 11 has also a large field of application in gaseous reactions. The discussion of this application will be found in my "Thermodynamische Vorlesungen über Technische Gasreaktionen."

Techn. Hochschule, Karlsruhe, Germany.

Electric Induction Furnace for Making Steel.

Mr. Viktor Engelhardt, chief chemist of the Siemens & Halske Co. in Berlin, made a visit to Sweden towards the end of last year to study and report upon the electric manufacture of steel in the induction furnace, as operated at Gysinge, Sweden, and his observations and conclusions are embodied in a voluminous report published in *Stahl und Eisen* of Feb. 1, 15 and March 1. Since the induction furnace has been so often described and noticed in these columns (Vol. I, p. 141, 283, 376, 462, 526, 546; Vol. II, p. 479; Vol. III, p. 134), only new details will be mentioned in the following:

The patents of Kjellin for the induction furnace are now owned by the Metallurgiska Patent Aktiebolaget, in Stockholm (which also owns the patents for Groendal's process of treating iron ores by means of magnetic separation and briquetting). This company has given licenses to the new owner of the Gysinge plant, to a "prominent English steel works," and to a French firm (at Voiron), which has already a 61-kw furnace in operation. The Siemens & Halske Co. have acquired the exclusive rights for Germany and Austria-Hungary.

At Gysinge, a 58-kw Kjellin furnace, finished in November, 1900, had an output of 600 to 700 kg steel in 24 hours, thus requiring 2140 kw-hours per ton of steel.¹ Later on a 170-kw furnace has been installed, and the results in the tests given in the following refer to this furnace.

The construction of this furnace was already shown in the

¹ In this article one ton means metric ton = 1000 kg = 2204 pounds.

drawings on page 577 of our first volume. That vertical portion of the magnet core which is inside the primary coil carrying the alternating current has not an exactly square cross-section, but the four corners of the square are taken off to provide vertical air chambers for cooling purposes.

The cross-section of the chamber, containing the metal, with its lining, is shown in Fig. 1. It will be seen that the cross-section of the metal chamber is essentially a rectangle with a rounded bottom. This is the only detail of construction which has not given complete satisfaction in practice. The trouble experienced is that the opening for charging is too narrow, and that it is difficult to repair the linings of the vertical walls of this chamber. It is, therefore, intended to use inclined walls in future, so that the chamber has a larger cross-section at the top than at the bottom. The cross-section of the chamber will, therefore, be in future more of the form of a triangle, the bottom being again rounded.

The first lining used in Gysinge was a silica lining, but this was not durable, and gave up too much silicon to the steel. It has, therefore, been replaced by a basic magnesite lining, stamped in a thickness of 300 mm.; 500 kilograms of sintered magnesite are mixed with 10 kilograms of finely ground caustic magnesite; then 40 kilograms of Holland clay are made up to a paste with water, and the whole is then mixed and stamped in. The entire furnace lining requires 2700 kilograms of sintered magnesite and the corresponding additions. Details of the cost² are given as follows:

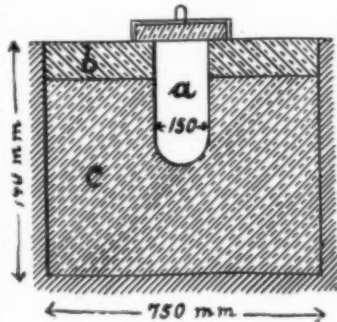
First cost of lining.....	\$121.86
Cost of repairs during twelve weeks.....	37.98
Removal of the lining.....	13.40
Total	\$173.24

On this one lining 285 tons of steel were made during a twelve-weeks run, making a total cost of lining amounting to 6.18 cents per ton of steel.

The operation is as follows: The charge being made up, it is placed behind the furnace, and as soon as the previous charge is run out the new one is put in, in two lots, the second half following 60 minutes after the first, and each charging consuming 15 minutes. The whole charge is molten in about two hours from the start, and a sample is then taken for carbon by the color test. The carbon being regulated to the desired point, the heat is increased, and 10 to 15 kg of 12 per cent ferro-silicon are added about 15 minutes before tapping. On tapping, if the steel is not perfectly quiet, a slight amount of aluminium (about 50 grams for a whole charge) is put into the metal in the casting ladle, and the steel is then cast into ingots of 85 to 200 kgs each. In the presence of Mr. Engelhardt, eight ingots were obtained from one charge, namely, four ingots each of 85 kg, three each of 100 kg, and one of 200 kg; the capacity of the furnace was 1350 kg, the total metal tapped 840 kg, and the duration of the operation 4 hours.

According to the record book of the plant an average of about 5000-kg ingots were cast in 24 hours, with an average

² The original figures in Mr. Engelhardt's report are given in Swedish crowns. In changing the figures to United States money, 1 crown is here assumed equals \$0.268, which is the figure used by the United States Treasury Department in 1902 for estimating the value of all foreign merchandise imported into this country.



power of 167.1 kw, corresponding to a consumption of 802 kw-hours per ton of steel. As an average of forty-eight further charges in 8 days at 170 kw, with a 4-hour duration of each charge, a power consumption of 770 kw-hours per ton is recorded. These figures refer to cold-starting materials. When 650 kg molten pig iron were introduced into the empty furnace and 1,300 kg cold pig iron and scrap were added, the power consumption was 650 kw-hours per ton of steel.

The figures 800 kw-hours for cold-starting materials and 650 kw-hours for molten starting materials are plotted in Fig. 2 for the 170 kw-furnace. These two figures therefore represent experimental results. The diagram shows how the power consumption per ton of steel depends upon the size of the furnace. The results obtained with the 61-kw furnace in France are also used in the diagram, the figure being 1220 kw-hours per ton. From the report of Mr. Engelhardt it would seem that these three figures for power consumption are the only ones in the diagram which were found by experiments, while the power consumption for larger sizes of furnaces are apparently estimated. It seems that a 736-kw furnace is in course of construction, and that for this furnace, with a capacity of 3740 kg and 2000 kg of tapped metal, the power consumption per ton of steel is estimated as 590 kw-hours when the starting material is cold pig iron, the daily output being 30 tons. If molten pig iron is the starting material the daily output is estimated to increase to 36 tons, corresponding to 490 kw-hours per ton. It is, of course, evident that the losses due to radiation and convection of heat (in percentage of the total power) are reduced by increasing the size of the furnace.

The figures for the power consumption, amounting to 489 or 331 kw-hours per ton with cold or molten starting materials respectively, are theoretical figures of Mr. Engelhardt, their calculation being given in detail in the original report.

If pig iron and ore are treated the power consumption is higher, and amounts in the average to 1200 kw-hours.

Mr. Engelhardt gives a great many tables giving the complete record of several charges carried out in his presence. Two of them, representing two different types of treatment, may be mentioned here. First, the "scrap process." The whole operation, using 400 kg of iron and 425 kg of open-hearth steel scrap, lasted 3 hours and 21 minutes. Tapping lasted 6 minutes. The scrap was introduced in two portions, 200 kg being introduced 10 minutes after starting and 225 kg 1 hour after starting; 16.5 kg of 98 per cent Cr were introduced 2¼ hours after starting, 10 kg of 12 per cent ferro-silicon 3 hours after starting. The total electrical energy used was 567.4 kw-hours. The analysis of the steel gave 1.71 per cent C, 0.13 Si, 0.30 Mn and 2.35 Cr.

As an example of the "ore process" the following run may be given: There were charged 850 kg of pig iron, and at intervals during 4 hours 85 kg of iron ore briquettes (introduced in four different portions), and 11 kg of limestone (in six portions), while just before tapping, nearly 6 hours after starting, 15 kg of 12 per cent ferro-silicon and 2 kg of 85 per cent ferro-manganese were added. A total of 1002.90 kw-hours was consumed in this run. The tapping lasted 5 minutes and the analysis of the steel gave:

	%C	Si	Mn	P
Beginning of tapping.....	1.19	0.17	0.24	0.102
Middle of tapping.....	1.11	0.18	0.25	0.086
End of tapping.....	1.18	0.13	0.24	0.150

The raw materials used in commercial work at Gysinge are given in Table I.

We may again return to Fig. 2 and Mr. Engelhardt's theoretical calculations. From the data obtained, Mr. Engelhardt calculates that a furnace of 1000 horse-power (736 kw) should have a capacity of 3740 kg and run out regularly 2-ton charges

TABLE I.

RAW MATERIALS USED IN GYSINGE.

Pig Iron, Dannemora C	4.5%	For making carbon and
Si	0.08	chrome steel by the "scrap
S	0.015	process."
P	0.018	
Mn	1.00	
Cu	0.015	
As	0.035	
\$22.00 per ton in Gysinge.		
Guldsmedhütte C	4.0%	For the "ore process" since
Graphite	1.16	richer in P.
Si	0.73	
P	0.051	
Mn	0.11	
Scrap from basic open-hearth		For making carbon and
furnace in Hohendahl. Assumed 0.1%.		chrome steels by "scrap
		process."
\$25.50 to \$26.80 per ton in Gysinge.		
From their own forge in Gysinge. Assumed 1.0%.		For making carbon and
		chrome steels by "scrap
		process."
Assumed \$26.80 per ton.		
Ferro-silicon made in blast		In every charge before tap-
furnace; 12% Si.		ping, to prevent blowholes.
\$30.80 per ton in Gysinge.		
Ferro-manganese made in blast		In "ore process" added at the
furnace; 85% Mn.		end of the charge.
Ferro-chrome, manufacturer		For making chrome steel.
unknown; 66% Cr, 8% C.		
Chromium, probably made by		For making chrome steel.
aluminothermic process; 98% Cr.		
Ore briquettes from Groendal	60 to 65% Fe.	For "ore process."
process of magnetic separation and briquetting.		
	69.5% Fe.	To be used later
		in "ore process." Was not
		yet available in Gysinge in
		sufficient quantity.
Assumed price \$4.30 to \$4.55 for 70% Fe free port.		

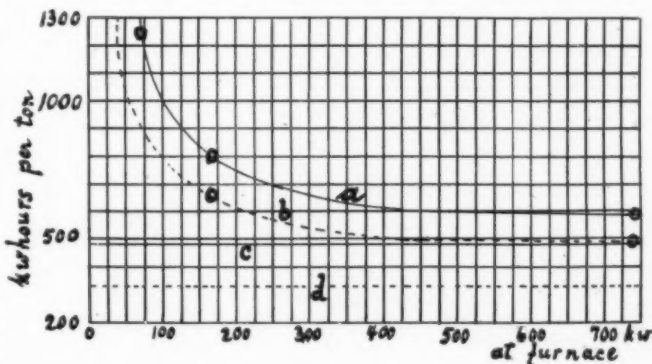


FIG. 2.—INFLUENCE OF SIZE OF FURNACE ON POWER CONSUMPTION.

each 1½ hours when using cold pig iron, or each 1-1.3 hours using melted pig iron, with a total output of 30 or 36 tons per day, using respectively 590 or 490 kw-hours per ton of steel. "Since there are theoretically needed 489 and 331 kw-hours per ton, in the two cases," the efficiencies to be attained with this size of furnace are 83 and 68 per cent respectively, and it is evident from the curves that larger furnaces would not attain any greater efficiency.

The abstractor calls attention to the fact that Mr. Engelhardt has calculated all his efficiencies too high; that the heat on the melted steel running from the furnace would not be over 340 calories per kilogram of steel at the most, whereas, it has been

assumed as 680 calories in his calculations, and the efficiencies stated as attained are to be at least cut in half. There is, however, a comforting compensation in the fact that this criticism also halves the heat or power stated as theoretically necessary to melt the steel, and so gives to the designers and operators of electric furnaces a hundred per cent greater chance for improvement and final economy of power than Engelhardt's figures promised.

At Gysinge a 300-horse-power turbine is directly coupled with a dynamo giving single-phase alternating-current of 3000 volts at 15 periods per second. The phase difference between the e.m.f. and the current changes in different runs with the quantity of materials treated, and it also changes in a single run according to the amount of material which has been charged into the furnace. With small charges (1350 kg) the power factor (cosine of the phase difference) is up to 0.80; with larger charges (like 1800 kg) it decreases in the average to 0.68.

Concerning the waste, it is stated that from the 23d to 28th of October 27,069 kg raw materials were charged into the furnace, and 25,131 kg steel were obtained, while there were at least 400 kg left in the furnace, so that the difference of 27,069 and 26,531, that is, 538 kg, or 2 per cent, represented the loss in this period. Mr. Engelhardt considers a 2 per cent waste as a fair average.

In the "scrap process" there is no possibility of removing certain elements which are not wanted. The process is of such a nature that whatever is introduced into the furnace is found afterwards in the product. The only exception is carbon. At Gysinge it is customary to calculate the amount of carbon introduced into the charge by the following method: From the amount of carbon calculated from the carbon contained in the metal left in the furnace, and from the new raw materials, a certain empirical percentage is taken off to account for the burning of carbon in air during charging, or by the oxygen contained in the raw materials, etc. This constant percentage is between 0.4 and 0.5 per cent, and is assumed in general as 0.45 per cent, while the exact adjustment is made by proper additions of pig iron, or ore, after the content of carbon has been determined by the color test, as follows:

Scrap Process.—The metal left in the furnace from the previous charge contains 1 per cent carbon, the next charge is desired also to contain 1 per cent carbon.

Old Charge:	Kg C.
400 kg steel at 1% C.....	4
New Charge:	
300 kg pig iron at 4.5% C.....	13.5
500 kg wrought iron at 0.1% C.....	0.5
75 kg scrap at 1% C.....	0.75
1275 kg, total	18.75

$18.75 \div 12.75 = 1.47\%$ C, subtracting 0.47% yields 1% C.

Mr. Engelhardt gives numerous mechanical tests of Gysinge steel. Some of them, made in the Institute of Technology in Stockholm, and on material showing less than 0.02 per cent P, 0.011 per cent S, and with carbon from 0.18 up to 0.91 per cent, the following figures were obtained:

Elongation, 9.0 to 23.8 per cent.

Limit of proportionality, 30.2 to 44.3 kg per sq. mm.

Elastic limit, 33.4 to 50.7 kg per sq. mm.

Tensile strength, 43.3 to 97.6 kg per sq. mm.

The quality is, therefore, close to the best of crucible steel.

Mr. Engelhardt then makes some comparisons of the induction furnace process, first with the crucible steel process and then with the open-hearth process. In comparison with the crucible steel process he concludes that the Kjellin furnace produces a steel equal in quality to crucible steel, but is able to furnish it at a lower cost of production. Since this has been pointed out repeatedly in our columns, and since no new

arguments are brought forward, this reference may suffice.

As far as concerns competition with the Siemens-Martin open hearth steel, Mr. Engelhardt assumes an average cost of \$18.75 to \$20.00 per ton for the basic process, and a cost of \$21.25 to \$22.00 per ton for the acid process. With a 736-kw electric induction furnace he estimates a total cost of \$17.85, or \$17.01 per ton, with cold or molten pig iron respectively as starting material. The only big item in this cost, which depends essentially on local conditions, is the cost of power; this is based on the assumption of 0.5 cent per kw-hour. Mr. Engelhardt, therefore, concludes that the electric induction furnace can compete successfully with the open-hearth furnace as long as the kw-hour does not cost more than 0.5 cent.

The Sources of Raw Materials for Niagara's Electrochemical Industries.

By MORRIS M. GREEN.

The development of hydroelectric power at Niagara Falls for electrochemical plants brings with it sundry interesting points regarding the cost of the various raw materials which are brought there for manipulation.

Among the most prominent of the materials are salt, used for making caustic soda in the Acker and Castner processes; coke, used in the graphite, carborundum, siloxicon and carbide industries; pure glass sand, used for graphite, siloxicon and carborundum manufacture; anthracite coal, used for making graphite; lime for bleaching powder and carbide. The most convenient sources and the approximate cost of these materials will now be discussed in detail.

SALT.

Fine beds of rock salt, suitable for manufacturing the purest evaporated salt, by first dissolving the rock salt, then evaporating by steam heat, are found in western New York near Warsaw, Retsof, and other towns, in the Genesee Valley.

There are two well-known methods of evaporating the salt from the brine, the first by "grainers" or vats open to the air, which produce a coarse salt; the other, a vacuum process in closed vacuum pans, producing a fine grained salt, at reduced consumption of fuel. With bituminous coal of fair quality, the relative fuel required per ton of salt, by the two processes is about as follows:

"Grainer" process	1300 pounds per ton of salt
Vacuum pan process..	900 pounds per ton of salt

Associated with chloride of sodium in the rock salt, are small quantities of sulphate of lime and chlorides of calcium and magnesium, all of which linger as impurities in the evaporated salt. Treatment of the brine with carbonate of soda will throw down carbonates of these bases, leaving soluble salts behind. This is desirable for some electrolytic processes, but also raises the cost of the salt.

Ordinary evaporated salt may be said to contain 98½ chloride of sodium.

With coal at \$2.50 per ton, the cost per ton, f. o. b. works, of salt may be said to run about as follows, including all expenses:

Grainer salt	\$2.12 per ton
Vacuum salt	1.70 per ton

The freight rate on salt per ton from the Warsaw district to Niagara Falls is 70 cents per ton for an 80-mile haul.

At Cleveland, Ohio, salt can be evaporated with coal costing not over \$1.25 per ton, but the distance to Niagara Falls is 200 miles, and the freight rate is \$1 per ton.

COKE.

The coke used in the electric furnace plants must be as pure as possible, as impurities require consumption of power for their volatilization.

Ordinarily the best grade of massive Connellsville beehive coke is bought, averaging 90 per cent fixed carbon.

The selling price of Connellsville coke at the ovens for the five years ending 1903 has averaged about \$2.30. Adding freight charges on the 300-mile haul to Niagara Falls would bring the price to about \$4 delivered.

It is unfortunate that the large amount of breeze coke, produced by the by-product coke oven plants should be so high in ash as to debar it for electric furnace work, as it is sold cheaply to find a market.

GLASS SAND.

Large quantities of the purest kind of glass sand are used at Niagara Falls in the carborundum, graphite and siloxicon plants. As impurities are objectionable for the reason mentioned for coke, the sand is shipped great distances. At one time Niagara Falls was supplied with glass sand, averaging 99 per cent silica, from a district near Pittsburg, Penn., 300 miles from Niagara Falls. At present the Carborundum Company is shipping sand 600 miles, from Wedron, Ill., to Niagara Falls. This deposit of sand has great natural advantages for mining cheaply, and is unusually pure, testing 99½ per cent SiO₂. This sand has been delivered in Niagara Falls for \$3.50 per ton, three quarters of this being transportation charges.

ANTHRACITE COAL.

This material, used in egg size for making artificial graphite, by the Acheson process, costs about \$4 a ton at the mines in the Lehigh Valley. Its cost at Niagara Falls, including freight, fluctuates between \$6 and \$7 a ton.

This material, being controlled by railroad interests, must be bought in the market.

LIME.

As the purest lime is required for electrochemical industries, only certain quarries of limestone, averaging 99 per cent Ca CO₃ are available.

Pure limestone can be quarried for 40 cents a ton, where strata are thick and well exposed, and where quarrying is carried on on a large scale. If the stone is burned with coke for fuel, 200 pounds of coke per ton of stone, or about 400 pounds per ton of lime, are required. If bituminous coal is used, 500 pounds per ton of lime are used.

Coal is much cheaper fuel; also lime can be burned with it without mixing with the ash of the coal, whereas if coke be mixed directly with the limestone, the ash of the coke tends to contaminate the lime.

The limestone for a ton of lime costs \$1, at a minimum; coal from 50 to 75 cents, so that lime seldom costs less than \$3 a ton, at a minimum. Four dollars a ton is a medium price, at the kiln.

Industrial Resistance Furnaces

(GIN, COLBY, KJELLIN.)

By F. A. J. FITZGERALD.

In attempting to discuss various forms of industrial furnaces of the resistance type we are at once met with the difficulty that, as a rule, details of the working of such furnaces, whether these are in the commercial or experimental stage, are lacking. If we desire to study the working of the blast furnace or the open-hearth furnace we have a voluminous literature, for these furnaces have reached a stage in their evolution where those who use them are not afraid of giving away something valuable if they say anything about them. A study of the patent literature in connection with electric furnaces shows that all experimenters are anxious to monopolize discoveries as far as possible, and in a new art like that of the working of electric furnaces such discoveries are naturally frequent. It thus happens that the experimenter, on overcom-

ing a difficulty in connection with his work, at once conceives the idea of monopolizing his discovery by an application to the patent office.

This attitude on the part of the experimenter is natural, and it is right that he should be rewarded for his work; but, unfortunately, it happens that from his lack of knowledge of the work of others he continually obtains patents on details that are already well known, or that have been patented by others, or in his anxiety to get a patent of some kind he obtains claims on well-known principles which cannot be regarded in any way as inventions or discoveries. Thus, on the one hand, we find a patent depending on the grading of carbon grains for the purpose of varying the resistivity of the resistor which they are intended to form; on the other hand, we find patents on obtaining increased temperature in a resistor by diminishing its sectional area or on adjusting the potential difference between the terminals of a furnace, so as to obtain a definite rate of generation of heat energy in the furnace.

Therefore, although some furnaces have been described with considerable detail as regards their construction and working, the majority of those in commercial use are more or less unknown quantities, and of those proposed for commercial use we have little information as to the experiments on which they are based. The only method of studying such furnaces is to apply general principles to their consideration, and attempt in that way to form some notion of their working.

GIN FURNACE.

Looked at from a purely ideal point of view the simplest form of electric furnace is that in which the current is passed

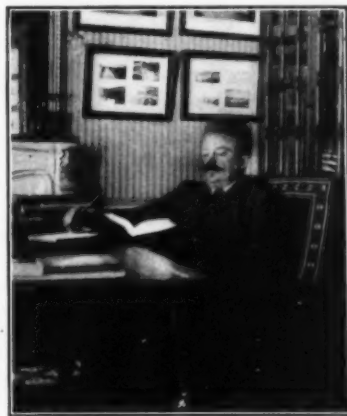
through the material to be heated, the heating effect being produced by the resistance of the substance itself, or, in other words, the furnace in which the charge constitutes the resistor. An example of this form of furnace is found in that proposed by Gin for the production of steel by melting cast iron and scrap or ore. The furnace is built by forming a long, narrow, shallow channel of refractory material such as chromite; the channel is filled with molten cast iron, and the current passed through the latter, the ends of the channel being supplied with water-cooled iron terminals for that purpose. A detailed description of the furnace may be found in an article by Bennie, *Electrochemical Industry*, Vol. II., No. 1, p. 20.

The heat generated in the channel of metal per second may be calculated from the equation:

$$JH = I^2R = \frac{E^2}{R} \quad (1)$$

Where J is the mechanical equivalent of heat, or the number of ergs given to a gram of water in raising it 1° C. in temperature, H, the number of calories (gram — degree C.), I is the current, E the potential difference between the ends of the channel and R the resistance, all expressed in absolute units. Since $J = 4.2 \times 10^7$, then if I, E and R are respectively amperes, volts and ohms, equation (1) becomes

$$H = \frac{I^2R}{4.2} = \frac{E^2}{4.2R} \quad (2)$$



G. GIN.

If the specific heat of the metal is c , its weight in grams is w , and the increase in temperature per second is θ , then

$$H = c w \theta \quad (3)$$

hence from (2) and (3)

$$c w \theta = \frac{I^2 R}{4.2} = \frac{E^2}{4.2 R} \quad (4)$$

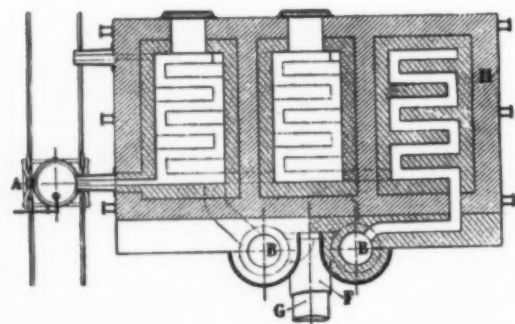
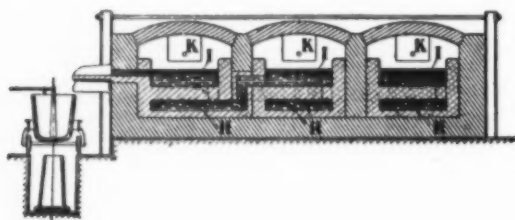
assuming that there is no loss of heat. Then the increase of temperature per second is

$$\theta = \frac{I^2 R}{4.2 c w} = \frac{E^2}{4.2 R c w} \quad (5)$$

If W is watts we also have this equation:

$$W = 4.2 c w \theta \quad (6)$$

Now, assuming that it is desired to heat 1 kg of iron from



FIGS. 1 AND 2.—GIN FURNACE.

0° to 1600° in one hour, and that the average value of the specific heat of iron is 0.3, then from (6)

$$W = \frac{4.2 \times 0.3 \times 1000 \times 1600}{3600} = 560$$

Therefore, provided there is no loss of heat, 560 watt-hours are required for 1 kg of iron, or 560 kw-hours for 1 metric ton.

The resistivity of molten iron, according to Gin, is 0.000175 ohm for a centimeter cube, therefore, if the length of the channel is l , its sectional area A , the current is 11,200 amps. 50 volts, and the density of the molten iron is 7, then we have

$$\frac{l}{0.000175} = \frac{50}{7 \times 11200}$$

$$7lA = 1000000$$

from which we get the following values:

$$l = 1910 \text{ cms}$$

$$A = 75 \text{ square cms.}$$

for a furnace that will fuse 1000 kg of iron per hour, assuming no loss of heat.

It is plain that in a furnace of such great length, compared with such a small cross section, the heat losses would be very great. To overcome this objection the channel is bent on itself several times, thus greatly diminishing the radiating surface.

In the furnace proposed by Gin it is not intended to start with cold iron in the channel, but to run in molten iron from a blast furnace or cupola, and then raise its temperature and treat it with ore or scrap iron, or both, so as to produce the desired grade of steel. It would be difficult to start a furnace of this kind using cold iron for the charge unless some device for regulating the voltage was used. Probably the best form in which to have the charge would be scrap iron in small pieces, which would offer numerous high-resistance points of contact; but a furnace charged in this way would certainly be difficult to manipulate. Once the furnace was filled with molten metal the practice probably would be to tap off only a portion of the charge and then add more iron to the furnace.

Here it may be well to consider a possible contingency, viz.: the freezing of the metal in the furnace. The resistivity of iron at ordinary temperatures may be assumed as 0.00001 ohm. Then, in order to use 560 kw for melting the frozen iron in the channel of the furnace, we have imagined we should require the volts (E) to be

$$E = \sqrt{\frac{0.00001 \times 1910 \times 560000}{75}}$$

and the current I would be

$$I = \frac{560000}{12} = 47000 \text{ amps.}$$

The electrical apparatus necessary for a contingency of this nature would be, at the present time, somewhat expensive.

In actual practice the current used in the furnace would be larger than appears in the calculations, since the assumption has been made that there are no heat losses. Even with the furnace built of dimensions given above and the channel so arranged as to avoid heat losses, the furnace would be highly unsatisfactory on account of the small sectional area of the channel. To shorten the channel and increase its cross-section it would be necessary to lower the volts and increase the amperes of the current, and then electrical difficulties will be encountered, for in using such a large

alternating current the self-induction of the furnace circuit becomes great and the power factor low.

Another objection that may be raised in connection with this furnace is the probable difficulty that would be experienced in keeping walls of the channel in repair. With an ideal refractory material such troubles might be avoided; but this material has still to be found.

In the simple channel furnace it would not be at all easy to carry out the necessary work of treating the molten iron with the materials required for producing the desired grade of steel. In a later design Gin has elaborated his furnace with the object of avoiding this objection, and the destructive action of the slag on the walls of the channel. This modified furnace is shown in vertical and horizontal section in Figs. 1 and 2. We have here three reservoirs I connected in series by the channels H , the terminals of the channels being at B, B . The heating of the charge is obtained by the passage of the current through the molten metal in the channels, while the reservoirs are used for charging the furnace and treating the iron to pro-



EDWARD A. COLBY.

duce the desired grade of steel. Thus, in treating the iron by the oxidation method a furnace with three reservoirs is used, the molten iron being charged into the first reservoir and drawn off from the third. In the first reservoir silicon and manganese may be removed in the second carbon and phosphorous, while in the third reservoir recarbonization is carried out and the steel tapped off. As the metal is drawn off from the third reservoir the metal which has been heating in the channel between the second reservoir and the third flows into the third reservoir, while the contents of the second reservoir flows into the channel. Meanwhile the contents of the other channels and reservoirs are flowing through the system in a similar manner.

Ignoring the objections that may be raised on *a priori* grounds to this type of furnace for smelting iron, it must be admitted that the ideal method of heating any substance by an electric current is to pass the current through the substance itself. But until some definite information on the actual working of such a furnace is obtained it would be well to suspend judgment thereon.

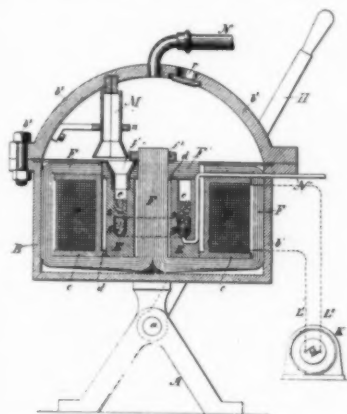


FIG. 3.—DIAGRAM OF KJELLIN FURNACE.
(FROM PATENT 428,552, MAY 20, 1890.)

THE COLBY AND KJELLIN FURNACES.

The furnace patented by Colby¹ and the similar furnace recently worked out by Kjellin, in Sweden,² heat the material under treatment by passing the current through it directly. In these furnaces the heated material forms the secondary circuit of a transformer, and, consequently, the use of terminals is avoided. In the manufacture of steel in the electric furnace the use of carbon terminals in contact with the molten steel must be avoided, and thus various devices, with this end in view, are employed. In the Gin furnace water-cooled steel terminals are used; but obviously doing away with terminals altogether, as in the Kjellin and Colby furnaces, is best. In this furnace, as in the Gin furnace, it is necessary to use a channel of considerable length as compared with the cross-section, and the difficulty of obtaining a suitable refractory material for its construction probably arises. A more serious disadvantage is found in the low power factor of the apparatus. Thus, in the case of the 225-kw

furnace shown to the Canadian commission the power factor was found to be about 0.6, and if it was desired to use larger furnaces the power factor would be still less. To correct this objectionable feature to a certain extent the frequency of the current may be decreased.

In spite of these obvious objections, the Kjellin furnace has been worked with considerable success and excellent steel produced by it. This furnace would compete with the ordinary crucible furnace for steel, and the very poor efficiency of the latter is well known, so that the induction furnace might be used with important economic advantages.

Metallurgical Calculations.—VI.

By J. W. RICHARDS, PH.D.

Professor of Metallurgy in Lehigh University.

In several of the preceding installments we have given the heats of formation of alloys and compounds and the thermodynamics of the elements. Before passing to the thermodynamics of alloys and compounds and problems involving their use, we will consider a few simple cases of the application of data so far given. Such include operations in which metals are melted or volatilized, or amalgams retorted. A few words may be in order, to clear the ground, regarding what is to be regarded as the efficiency of a furnace.

EFFICIENCY OF FURNACES.

Under this term we must distinguish a generic sense and a specific sense, the first referring to furnaces in which the object is to maintain a certain temperature for a certain time with the minimum consumption of fuel, the second, in which the object is to perform a certain thermal operation with the smallest consumption of fuel. In the first case, one furnace may be compared with another, and thus comparative efficiencies calculated; in the second case real or absolute efficiencies can also be calculated. A few examples will illustrate this difference, which is an essential difference as far as making calculations is concerned.

Cases of Specific Efficiency: Whenever it is desired to melt a metal for the purpose of casting it, a certain definite amount of heat must be imparted to the metal, and the ratio between this efficiently utilized heat and the heating power of the fuel consumed, is the efficiency of the furnace. If the furnace is electric the theoretical heat value of the electric energy used is the divisor. If, in addition to the heat required to raise the substances to the desired temperature, there is also heat absorbed in chemical reactions, this amount can be added in as usefully applied heat, and the sum of this and the heat in the final products be regarded as the total efficiently applied heat. If a blast furnace takes iron ore and furnishes us melted pig iron, the sum of the heat absorbed in the chemical decomposition of the iron oxide and the sensible heat in the melted pig iron is the efficiently applied heat, because it is the necessary theoretical minimum required; all other items³ are more or less susceptible of reduction, but these are necessary items and, therefore, measure the net efficiency. If my dwelling requires 200 cubic feet of hot air per minute at 150° F. to keep it at 65° F., while the outside air is at 0° F., the ratio of the heat required to warm the 200 cubic feet of air from 0° F. to 150° F., to the calorific power of the fuel used per minute, measures the *specific* efficiency of the "heater;" the question whether this amount of hot air keeps the temperature of the rooms at 65° F. is a question of the *general* efficiency of the construction of the house.

Cases of Generic Efficiency: Such are those in which practically all the heat generated eventually leaves the furnace by radiation or conduction, or useless heat in waste gases; this is the case when a certain temperature has to be continuously maintained for a given time, and where the *time* element is the



F. A. KJELLIN.

¹U. S. Patents, Nos. 428,378 428,379, May 20, 1890.

²*Electrochemical Industry*, Vol. 1., No. 11, p. 376.

controlling one, and not any definite amount of thermal work is to be done. Examples are numerous: An annealing furnace, where steel castings, let us say, are to be kept at a red heat for two days, or a brick kiln, where several days slow burning are required, or a puddling furnace, where the melted iron must be held one to two hours to oxidize its impurities. In all these cases we may say that one furnace keeps its contents at the right heat for the right time with so much fuel, another does the same work with 10 or 25 per cent less fuel, and is, therefore, 10 or 25 per cent more efficient; but we cannot, in the nature of the case, speak of the absolute or specific efficiency of the furnace, because there is no definite term, expressible in calories, to compare with the thermal power of the fuel.

In many cases the two efficiencies are mixed in the same process or operation, and then the calculation of absolute or specific efficiency can be made for that portion of the operation wherein a certain definite amount of thermal work is done. Thus, in an annealing kiln, 50 tons of castings may be brought up to annealing heat in 24 hours, starting cold, and the heat absorbed by the castings compared with the calorific power of the coal burnt during this period is a measure of the real efficiency of this part of the operation. During the rest of the operation, while the castings are simply kept at annealing heat, there can be no calculation of the absolute or specific efficiency of the furnace, because one of the terms necessary for the comparison has disappeared; in that part of the process we can only speak of relative efficiency compared to some other furnace doing a similar operation.

It goes, almost without saying, that we can, of course apply the conception of efficiency in its relative or general sense to the whole operation or to any part of it.

Problem 6.

The Rockwell Engineering Co. state in their current advertisements that their regenerative oil-burning furnace melts 100 pounds of copper with the consumption of less than 1.5 gallons of oil. Assume that 1.5 gallons of oil is used, and that the copper is heated from 25° C. to melted metal 100° C. above its melting point.

Required: The "efficiency" of the furnace; i. e., its specific efficiency as calculated from the net heat utilized.

Solution: One gallon of fuel oil averages in weight 7.5 pounds, and its calorific power 11,000 Calories per kilogram, or 11,000 pounds Calories per pound. The calorific power of the fuel used in melting 100 pounds of copper is therefore:

Heat generated $11,000 \times 7.5 \times 1.5 = 82,500 \times 1.5 = 123,750$ pounds Calories.

The heat imparted to the copper is as follows, taking the data from Article V. of these calculations:

Heat in 1 lb. melted copper at melting point = 162 lb. Cal.
Heat in 1 lb. solid copper at 25° C. = 2 "

Heat required to just melt the copper = 160 "
Heat to superheat liquid copper 100° C.

$0.133 \times 100 = 13$ "

Total heat expended on each pound of copper = 173 "
Heat usefully applied per 100 pounds = 17,300 "

$$\text{Net efficiency of furnace} = \frac{17,300}{123,750} = 0.14 = 14\%$$

It is proper to remark that although this efficiency appears low, yet it is considerably greater than is attained in simple melting holes or wind furnaces, and yet the calculations show what a large margin for improvement and greater efficiency exists in even some of the best and relatively most efficient metallurgical furnaces.

Problem 7.

In the distillation of silver amalgam in iron retorts, 1000

kilos. of amalgam, containing 200 kilos. of silver, is retorted with the consumption of 550 kilos. of wood, the mercury vapor passes off at an average temperature of 450° C., and the silver is raised towards the end of the operation to 800° C., in order to expel the last of the mercury. Assume the calorific power of the wood 3000 Calories.

Required: The net efficiency of the furnace.

Solution: The heating power of the wood is $550 \times 3000 = 1,650,000$ Calories.

The heat utilized is that absorbed in separating the silver from the mercury, plus the sensible heat in the mercury vapor at 450°, plus the sensible heat in the silver at 800°. These are calculated as follows:

Heat to decompose amalgam = 2470 Calories per 108 kilos. of silver = $200 \times (2470 \div 108) = 200 \times 238 = 47,600$ Calories.

Heat in silver at 800°, using Pionchon's formula, is 800 $[0.05758 + 0.0000044 (800) + 0.000000006 (800)^2] \times 200 = 10,390$ Calories.

Heat in 800 kilos. of mercury vapor at 450° is

(a) heat to boiling point (Naccari)	
$356 [0.03337 - 0.0000275 (356) - 0.000000667 (356)^2] \times 800$	= 14,699 Cal.
(b) heat to vaporize 77.5×800	= 62,000 "
(c) heat in vapor at 450° = $0.025 \times (450 - 356) \times 800$	= 1,880 "
Total	= 78,579 "

Heat usefully applied:

In decomposing amalgam	= 47,600 Cal.
In mercury vapor, as sensible heat	= 78,579 "
In silver, as sensible heat	= 10,390 "
Total	= 136,569 "

$$\text{Efficiency of furnace} = \frac{136,569}{1,650,000} = 0.083 = 8.3\%$$

Problem 8.

In a zinc works, impure zinc is refined by redistillation in fire-clay retorts, a bank of retorts distilled 970 kilos. of zinc with the expenditure of 912 kilos. of small anthracite coal. Assume that the zinc vapors pass out of the muffles at the boiling point (930°).

Required: (1) The net efficiency of the furnace.

(2) The electrical power which would be required, in horse-power-hours, to do the same work, assuming the heating efficiency of the electric furnace is 75 per cent.

Solution: (1) The small anthracite may be assumed to have a calorific power of 7850 Calories; therefore, the total heat which should be developed is $7850 \times 912 = 7,159,200$ Calories. The heat in 1 kilo. of zinc in the state of vapor at its boiling point can be calculated from the thermophysical data supplied for zinc as:

(a) In solid zinc to melting point (420°)	47.34 Cal.
(b) Latent heat of fusion	22.61 "
(c) Heat in melted zinc to boiling point (930°)	65.05 "
(d) Latent heat of vaporization	370.15 "
Total.	505.15 "

Heat required for 970 kilos. = 489,995. "

$$\text{Efficiency of furnace} = \frac{489,995}{7,159,200} = 0.068 = 6.8\%$$

(2) One electric horse-power-hour	= 6440 Cal.
Efficiently applied heat = $644 \times .75$	= 483.0 "

$$\text{Electric horse-power-hours required} = \frac{489,995}{483} = 1013 \text{ E.H.P. hours.}$$

One metric ton of zinc requires $\frac{1013}{0.970} = 1044$ E.H.P. hours.

Cost of power, at \$20.00 per E.H.P. year = $\frac{20.00}{8766} \times 1044 = 0.00228 \times 1044 = \2.38 .

This cost of electric power would replace the use of $\frac{0.912}{0.970}$ metric tons of small anthracite, equal to a cost of \$2.53 for electric power sufficient to replace a metric ton of coal for this purpose.

Many other examples could be given of the technical use of the thermophysical data concerning the elements, but the problems given illustrate the methods of calculation.

When one is acquainted with some of the ordinary metallurgical operations, such as melting and distilling the metals, it is surprising to notice how little is known or thought of the efficiency or lack of efficiency of the furnaces used. One man melts 100 pounds of metal by the use of 150 pounds of coal, he builds a new furnace and does it more cheaply by using 100 pounds of coke, which is certainly relatively more efficient; but it is seldom that the operator knows that in one case he is getting probably only 7 per cent efficiency from his fuel and in the other case only 10 per cent. It is the knowledge of these absolute efficiencies which tells the practical man just what he is accomplishing, and shows him how much room there still remains for improvement.

THERMOPHYSICS OF ALLOYS.

There does not exist, in technical literature, much data of this nature concerning alloys. There is here a wide and interesting field for metallurgical research, whose cultivation would yield results both of high practical and high theoretical interest, and yet it is comparatively untouched. What is wanted is complete data concerning the specific heat of solid and liquid alloy, and latent heat of fusion. These, combined with the determination of the heat evolved in the alloying, would furnish a sound basis for a practical theory of alloys, besides enabling workers with these alloys to control the efficiency of their furnaces and, in general, to know with scientific exactness what they are accomplishing.

ALLOYS OF TIN AND LEAD.

Per Cent of Tin.	Sm.	Latent Heat of Fusion.
4.8 (Pb ¹⁰⁰ Sn)	5.5 (Mazotto)
10.2 (Pb ⁸⁹ Sn)	8.0 at 307° (Spring)
12.5 (Pb ⁸⁷ Sn)	8.3 at 292° (Spring)
16.0 (Pb ⁸⁴ Sn)	9.1 at 289° (Spring)
22.2 (Pb ⁷⁷ Sn)	9.5 at 270° (Spring)
		7.9 (Mazotto)
36.3 (PbSn)	0.04073 (12° — 99°) (Regnault)	11.6 at 241° (Spring) 9.4 (Mazotto)
50.0 total heat to 0° in 1 kilo. melted metal		18.0 from 202° (Ledebur)
53.3 (PbSn ⁴⁶)	0.04507 (10° — 99°) (Regnault)	10.5 at 197° (Mazotto)
63.1 (PbSn ³)	15.5 at 179° (Spring)
69.5 (PbSn ²)	17.0 at 188° (Spring)
83.0 total heat to 0° in 1 kilo. melted metal		21.5 from 205° (Ledebur)
90.1 (PbSn ¹⁰)	12.9 (Mazotto)

ALLOYS OF TIN AND BISMUTH.

Per Cent of Tin.	Sm.	Latent Heat of Fusion.
67 (Bi ³³ Sn)	11.4 Cal. (Mazotto)
22.1 (Bi ⁷⁷ Sn)	11.2 Cal. (Mazotto)

36.2 (BiSn)	0.0400 (20° — 99°) (Regnault)	11.6 Cal. (Mazotto)
53.1 (BiSn ⁴⁶)	solid, 0.0450 (Regnault) liquid, 0.0454 (146° — 275°) Person	11.6 Cal. (Mazotto)
69.1 (BiSn ³)	11.1 Cal. at 140° (Mazotto)
82.7 (BiSn ²)	12.6 Cal. (Mazotto)
90.1 (BiSn ¹⁰)	12.8 Cal. (Mazotto)

ALLOYS OF TIN AND ZINC.

Per Cent of Tin.	Sm.	Latent Heat of Fusion.
78.4 (ZnSn ²¹)	23.5 Cal. (Mazotto)
92.7 (ZnSn ⁷)	16.2 Cal. at 197° (Mazotto)
95.6 (ZnSn ⁴)	16.3 Cal. (Mazotto)
97.3 (ZnSn ³)	15.1 Cal. (Mazotto)

ALLOYS OF TIN AND COPPER.

Bell metal (20 per cent tin) Sm (14° — 98°) = 0.0862 (Regnault).

Bronze (15 per cent tin).

Total heat in melted metal (to 0°) = 130 Cal. (Ledebur).

If strongly superheated = 143.5 Cal. (Ledebur).

ALLOYS OF TIN AND ANTIMONY.

Britannia metal (90 per cent tin) requires to melt it, starting cold, 28.0 Calories per kilogram (melting point 236°); with 82 per cent of tin, 25.7 Calories; melting point 205° (Ledebur).

ALLOYS OF TIN, BISMUTH AND ANTIMONY.

BiSn²Sb (bismuth 34.3, tin 41.9, antimony 23.8 per cent).

Sm (15° — 100°) = 0.0462 (Regnault).

ALLOY OF TIN, BISMUTH, ANTIMONY AND ZINC.

BiSn²SbZn² (bismuth 29.8, tin 34.0, antimony 17.3, zinc 18.9 per cent).

Sm (15° — 100°) = 0.0566 (Regnault).

ALLOYS OF LEAD AND BISMUTH.

Per Cent of Lead.	Sm.	Latent Heat of Fusion.
11.1 (PbBi ⁸⁸)	10.2 (Mazotto)
33.2 (PbBi ⁶⁶)	6.4 (Mazotto)
39.9 (Pb ⁶⁰ Bi ⁴⁰) solid,	0.03165 (16° — 99°) Person	
	liquid, 0.03500 (144° — 358°) Person	
42.7 (Pb ⁵⁷ Bi ⁴³)	4.7 at 127° (Mazotto)
49.9 (PbBi)	4.0 (Mazotto)
66.6 (Pb ³³ Bi)	3.6 (Mazotto)
88.8 (Pb ¹¹ Bi)	4.9 (Mazotto)

ALLOYS OF LEAD AND ANTIMONY.

With 63.0 per cent of lead, Sm (10° — 98°) = 0.0388 (Regnault).

With 82.0 per cent of lead, heat in 1 kilo. melted metal = 15.6 Calories (Ledebur).

With 90.0 per cent of lead, total heat in 1 kilo. melted metal = 13.8 Calories (Ledebur).

ALLOYS OF LEAD, TIN AND BISMUTH.

D'Arcet's Alloy, containing 32.5 lead, 18.5 tin, 48.7 bismuth:

Sm solid (5° — 65°)	= 0.0372 (Mazotto)
Sm solid (12° — 50°)	= 0.049 (Person)
Sm solid (14° — 80°)	= 0.060 (Person)
Sm liquid (107° — 136°)	= 0.047 (Person)
Sm liquid (120° — 150°)	= 0.0399 (Mazotto)
Sm liquid (136° — 300°)	= 0.0360 (Person)
Latent heat of fusion	= 5.96 Cal. at 96° (Person)
	= 5.77 Cal. at 99° (Mazotto)

Rose's Alloy, containing 24.0 lead, 27.3 tin, 48.7 bismuth:

Sm solid ($5^{\circ} - 65^{\circ}$)	= 0.375 (Mazotto)
Sm fluid ($119^{\circ} - 338^{\circ}$)	= 0.0422 (Person)
Latent heat of fusion	= 6.85 Cal. at 99° (Mazotto)

Fusible Alloy, containing 31.8 lead, 36.2 tin, 32.0 bismuth:

Sm solid ($18^{\circ} - 52^{\circ}$)	= 0.0423 (Person)
Sm solid ($11^{\circ} - 98^{\circ}$)	= 0.0448 (Regnault)
Sm fluid ($143^{\circ} - 330^{\circ}$)	= 0.0460 (Person)
Latent heat of fusion	= 7.63 Cal. at 145° (Person)

Wood's Alloy, containing 25.8 lead, 14.7 tin, 52.4 bismuth, 7 cadmium:

Sm solid ($5^{\circ} - 50^{\circ}$)	= 0.0352 (Mazotto)
Sm fluid ($100^{\circ} - 150^{\circ}$)	= 0.0426 (Mazotto)
Latent heat of fusion	= 7.78 Cal. at 75° (Mazotto)

Lipowitz's Alloy, containing 25.0 lead, 14.2 tin, 50.7 bismuth, 10.1 cadmium:

Sm solid ($5^{\circ} - 50^{\circ}$)	= 0.0345 (Mazotto)
Sm fluid ($100^{\circ} - 150^{\circ}$)	= 0.0426 (Mazotto)
Latent heat of fusion	= 8.40 Cal. at 75° (Mazotto)

ALLOYS OF COPPER AND ZINC.

Red Brass	Sm at 0° = 0.0899 (Lorenz)
	Sm at 50° = 0.0924 (Lorenz)
(Copper 85%)	Sm at 75° = 0.0940 (Lorenz)
Yellow Brass	Sm at 0° = 0.0883 (Lorenz)
	at 50° = 0.0922 (Lorenz)
(Copper 65%)	at 175° = 0.0927 (Lorenz)

Heat in 1 kilo. of melted, somewhat superheated, brass = 130 Calories (Ledebur).

ALLOYS OF COPPER, ZINC AND NICKEL.

German Silver:

$$(74\text{Cu. } 20\text{Zn. } 6\text{Ni}) \text{ Sm}(0-1) = 0.0941 + 0.0000053t \quad (\text{Tomlinson})$$

ALLOYS OF COPPER AND ALUMINIUM.

$$\text{Copper } 88.7\% \text{ Sm } (20^{\circ} - 100^{\circ}) = 0.10432 \text{ (Luginin)}$$

ALLOYS OF SILVER AND PLATINUM.

$$\text{Silver } 66.7\% \text{ Sm } (0-1) = 0.04726 + 0.0000138t \quad (\text{Tomlinson})$$

ALLOYS OF MERCURY AND TIN.

HgSn (37.1% Sn)	Sm ($-30^{\circ} - 15^{\circ}$) = 0.04083 (Schüz)
	($-25^{\circ} - 15^{\circ}$) = 0.04218 (Schüz)
	($-22^{\circ} - 99^{\circ}$) = 0.07204 (Regnault)
HgSn (54.1% Sn)	Sm ($-25^{\circ} - 99^{\circ}$) = 0.06591 (Regnault)
HgSn (74.7% Sn)	Sm ($-16^{\circ} - 15^{\circ}$) = 0.05039 (Schüz)

ALLOYS OF MERCURY AND LEAD.

PbHg (50.9% Pb)	Sm ($-69^{\circ} - 20^{\circ}$) = 0.03458 (Schüz)
	Sm ($-23^{\circ} - 99^{\circ}$) = 0.03827 (Regnault)
PbHg (67.4% Pb)	Sm ($-72^{\circ} - 20^{\circ}$) = 0.03248 (Schüz)

ALLOYS OF CADMIUM AND TIN.

$$\text{CdSn}^2 (67.8\% \text{ Sn}) \text{ Sm } (-77^{\circ} - 20^{\circ}) = 0.05537 \text{ (Schüz)}$$

whence we have $\text{Sm } (0-1) = 0.0557 + 0.00000366t \text{ (Schüz)}$

ALLOYS OF IRON AND CARBON.

$$\text{Soft Steel } (0.15\% \text{ carbon}) \text{ Sm } (20^{\circ} - 98^{\circ}) = 0.1165 \text{ (Regnault).}$$

$$\text{Hard Steel } (1.00\% \text{ carbon}) \text{ Sm } (20^{\circ} - 98^{\circ}) = 0.1175 \text{ (Regnault).}$$

Total heat in 1 kilo. melted steel at 1350° = 300 Calories (Ledebur).

$$\text{Cast Iron } (4.0\% \text{ carbon}) \text{ Sm } (0 - 1200^{\circ}) = 0.175.$$

$$\text{Sm } (0-t^{\circ}) = 0.12 + 0.000046t$$

Total heat in 1 kilo. melted at 1200° = 245 Calories (Ledebur).

Total heat in 1 kilo coming from blast furnace = 250 to 325 Calories (Akermann).

Problem 9.

A steel-melting crucible contains 110 pounds of steel, which is melted in a wind furnace with the use of 150 pounds of coke. Assume the coke to be 50 per cent fixed carbon, and the steel to be superheated 100° C. above its melting point.

Required: The net efficiency of the furnace.

Solution: The calorific power of the coke may be assumed as 50 per cent that of pure carbon, and therefore:

$$= 150 \times (8100 \times 0.90) = 150 \times 7380 = 1,093,500 \text{ lb. Cal.}$$

Heat in steel at melting point:

$$110 \times 300 \text{ (Ledebur)} = 33,000$$

Heat to superheat 100° :

$$110 \times 100 \times 0.15 \text{ (assumed)} = 1,650$$

$$\text{Total} \quad 34,650 \text{ lb. Cal.}$$

$$34,650$$

$$\text{Efficiency of furnace} = \frac{34,650}{1,093,500} = 0.032 = 3.2\%$$

$$1,093,500$$

Problem 10.

A Siemen's regenerative furnace holds eighteen steel crucibles, each containing 100 pounds of steel. Assume that the efficiency of utilization of the heat for melting the steel is 5 per cent, and that the furnace is fed by natural gas, having a calorific power of 512-pound Calories per cubic foot.

Required: The number of cubic feet of natural gas required per furnace heat of eighteen crucibles = 1800 pounds of cast steel.

$$\text{Solution: Heat in steel} = 1800 \times 315 = 567,000 \text{ lb. Cal.}$$

$$\text{Heating power of gas required} = \frac{567,000}{0.05} = 11,340,000 \text{ lb. Cal.}$$

$$\text{Cubic feet of gas required} = \frac{11,340,000}{512} = 22,150 \text{ cubic feet.}$$

$$\text{Gas required per ton of steel, 2000 lbs.} = 24,610 \text{ cubic feet.}$$

$$\text{Cost of gas, at } \$0.08 \text{ per 1000 cubic feet} = \$1.97.$$

Problem 11.

In a malleable-casting foundry the pig iron is melted in a reverberatory air furnace, 3000 kilos. being melted in two hours by the combustion of 1200 kilos. of bituminous coal, having a calorific power of 8500 Calories.

Required: The melting efficiency of the furnace.

Solution: Calorific power of coal used:

$$1200 \times 8500 = 10,200,000 \text{ Calories.}$$

Heat in melted iron at foundry heat:

$$3000 \times 250 \text{ (Ledebur)} = 750,000 \text{ Calories.}$$

$$\text{Efficiency of furnace} = \frac{750,000}{10,200,000} = 0.0735 = 7.35\%$$

Problem 12.

In an iron foundry cupola 14 metric tons of pig iron are melted in one hour, using 1.5 tons of coke (90 per cent carbon). The gases passing away contain by volume CO 13 per cent, CO₂ 13 per cent, nitrogen 76 per cent, and leave the cupola at 500° C. The body of the cupola is 1.5 meters in diameter outside and 4 meters high.

Required:

- (1) The net melting efficiency of the cupola.
- (2) The proportion of the calorific power of the coke lost.
 - (a) By the sensible heat of the hot gases escaping.
 - (b) By the imperfect combustion of the coke.
 - (c) By radiation from bottom and walls of the cupola.
- (3) The amount of heat in Calories radiated, on an average, from each square meter of outside surface per minute.

Solution:

- (1) Calorific power of the coke:

$$1500 \times 0.90 \times 8100 = 10,935,000 \text{ Calories.}$$

Heat in melted iron:

$$14,000 \times 250 \text{ (Ledebur)} = 3,500,000 \text{ Calories.}$$

$$\text{Efficiency of melting} = \frac{3,500,000}{10,935,000} = 0.32 = 32\%$$

- (2, a) Weight of carbon escaping =
- $1500 \times 0.90 = 1350$
- kilos.

$$\text{Volume of CO and CO}^2 \text{ escaping} = \frac{1350}{0.54} = 2500 \text{ m}^3$$

(Because 1 m³ of either gas carries 0.54 kilos. C.)

$$\text{Volume of escaping gases} = \frac{2500}{0.13 + 0.13} = 9615 \text{ m}^3$$

$$\text{Volume of nitrogen (by difference)} = 7115 \text{ m}^3$$

Sensible heat of nitrogen and CO

$$(7115 + 1250) \times [0.303 (500) + 0.000027 (500)^2] = 1,323,760 \text{ Cal.}$$

Sensible heat of CO²

$$1250 \times [0.37 (500) + 0.00022 (500)^2] = 300,000 "$$

$$\text{Total sensible heat in gases} = 1,623,760 "$$

Proportion of calorific power of the fuel thus lost:

$$= \frac{1,623,760}{10,935,000} = 0.1485 = 14.85\%$$

- (2, b) Volume of CO escaping = 1250 m
- ³

$$\text{Calorific power of this gas} = 1250 \times 3062 = 3,827,500 \text{ Cal.}$$

Proportion of calorific power of the fuel thus lost:

$$= \frac{3,827,500}{10,935,000} = 0.35 = 35\%$$

- (2, c) Heat in pig iron = 3,500,000 Cal.

$$\text{Heat in waste gases} = 1,623,760 "$$

$$\text{Lost by imperfect combustion} = 3,827,500 "$$

$$\text{Accounted for} = 8,951,260 "$$

$$\text{Calorific power of the coke} = 10,935,000 "$$

$$\text{Difference, loss by radiation} = 1,983,740 "$$

Proportion of calorific power of the fuel thus lost:

$$= \frac{1,983,740}{10,935,000} = 0.1815 = 18.15\%$$

- (3) Area of bottom of cupola =
- $(1.5)^2 \times 0.7854 = 1.77 \text{ m}^2$

$$\text{Area of sides of cupola} = 1.5 \times 3.14 \times 4 = 18.85 \text{ m}^2$$

$$\text{Total radiating area} = 20.62 \text{ m}^2$$

$$\text{Heat radiated per sq. m. per hour} = \frac{1,983,740}{20.62} = 96,200 \text{ Cal.}$$

$$\text{Heat radiated per sq. m. per min.} = \frac{96,200}{60} = 1,603 "$$

In general, we may state that the net efficiency in melting, etc., of metals is very various, from, say, 2 or 3 per cent in a crucible steel-melting wind furnace to about 10 or 15 per cent in reverberatory furnaces, 20 to 30 per cent in regenerative open-hearth furnaces, 30 to 50 per cent in shaft furnaces, where material to be heated and fuel burned are in direct contact with each other, 50 to 75 per cent in steam boilers and hot-blast stoves, and 60 to 85 per cent in large electrical furnaces.

[The next instalment of these calculations will take up the thermophysics of compounds such as occur in metallurgical practice.]

The Electrochemistry of the Metallic Arc.

BY ISADOR LADOFF.

The object of this article is to review the literature treating of the metallic arc, scattered on the pages of various scientific and technical publications in the United States and foreign countries, and relate concisely the results of my own work in this field.

Berthold Monasch's book, "Der Electriche Lichtbogen," etc., published in 1904, by the Berlin firm of Julius Springer, was a great help to me owing to the excellent bibliographical data furnished by it.

Davy found that the metallic arc is not as luminous as the carbon arc. Grove claimed in 1840 that various metals produce arcs of different luminosity, and arranged the metals accordingly in series, in which each foregoing metal is alleged to produce a more brilliant arc than the next following metal, namely, K, Na, Zn, Hg, Fe, Sn, Pb, Cu, Ag, Au, Pt. Later investigators doubted the correctness of Grove's series. Guye and Monasch, for instance (Drud. Ann. 1, p. 702, 1900), found that an arc produced by Bi-electrodes with a current of 0.06 amps. could not be maintained for any considerable length of time. Arons had a similar experience with zinc electrodes. Wurts (Ann. Inst. El. Eng. 9, p. 102, 1893) found that no permanent arc can be maintained by zinc, bismuth and antimony, probably due to the insulating properties of the oxides of these "non-arcng" metals. As a rule, metallic arcs are unsteady and flickering, the metallic electrodes producing the arc being easily softened or melt and oxidize in the air. De La Rive noticed the formation of brownish-red oxides by the iron arc at the usual pressure in the air and dark oxides in a rarified atmosphere. Gold (Berichte d. Wiener K. Acc. d. Wiss. 104, IIa, p. 815, 1895) and Arons (Wied. Ann. 57, p. 199, 1896) confirmed these observations. According to Guye and Monasch (Ecl. El. 34, p. 305; 35, p. 18, 1903) aluminium showed the highest degree of oxidation in the arc. Lead sends out into the arc a whitish blue vapor, condensing into a powder of the same color on the inner surface of the globe surrounding the arc. At the same time the products of oxidation settle on the pencil in colored circles; a dark circle of sub-oxide of lead and a yellow of lead oxide are followed by a circle of brownish red color. An antimon arc behaves in about the same way and manner as a lead arc. A cadmium arc does not produce any white vapors, but other products of oxidation. Magnesium electrodes, similar to aluminium, produce in the arc a thick crust of insulating earthy oxides. Nickel electrodes cover under the action of the arc with a film of oxides. Copper electrodes, between which an arc is maintained for some time by a current of 6 amps., cover with a thick crust of black oxide. However, when the current does not exceed 0.06 amps. only a thin layer of black oxide is formed. Guye and Monasch noticed a blackish sublimation even on silver and platinum electrodes after an arc was kept up between them for some time.

In connection with the behavior of different metallic electrodes in the air it will be instructive to study their behavior in other gases.

Arons (Drud. Ann. 1, p. 700, 1900) generated nitrogen by carefully heating a concentrated solution of $(\text{NH}_4)_2\text{SO}_4$ and 2NaNO_3 , and dried the gas in which he produced metallic arcs. The most remarkable phenomenon observed by him was that no lasting arc could be maintained between silver electrodes with low-tension currents while employing the usual pressure in an atmosphere of nitrogen. Arons explains this phenomenon by the small affinity existing between silver and nitrogen. He advances the theory, that in order to keep up the arc some chemical combinations between the material of the electrodes and the surrounding atmosphere are essential or necessary. I. Thomson concluded from certain phenomena produced by discharges in gases, that the formation of certain chemical combinations precedes the passage of electricity from

the gases to metallic electrodes. Guye and Monasch produced an arc between silver electrodes with currents of high tension in an atmosphere of nitrogen containing about 3.50 per cent impurities. Owing to these impurities the experiment could not be pronounced as conclusive. Arons proved that some metals, as, for instance, aluminium and magnesium, produce nitrides in an atmosphere of nitrogen under the influence of the arc.

Grove succeeded, in 1840, to produce a carbon arc in an atmosphere of hydrogen, but failed to produce a lasting arc between metals in the same gas. Herwig (Pogg. Ann. 149, p. 523, 1873) and Stenger (Wied. Ann. 25, p. 31, 1885) repeated the experiment and confirmed Grove's conclusion. Arons used a current of 105 volts for the production of metallic arcs in hydrogen. He concluded that some metals do not arc at all in hydrogen, while others produce arcs only under special conditions. At a pressure exceeding 400 mm. Hg. column, no metals produce an arc in hydrogen.

Copper and aluminium easily producing an arc in nitrogen did not arc at all in hydrogen. Platinum and silver arcs demanded very high currents, endangering the electrodes. The same applied to iron, and especially to lead, due to the latter's melting point. Cadmium, zinc and magnesium showed in hydrogen the most favorable results.

Crew and Basquine (Proc. Am. Acad. 33, p. 335, 1897, 1898), who succeeded to produce in hydrogen arcs between iron, zinc and magnesium electrodes, call attention to the reducing effect of the gas on iron electrodes previously oxidized in the air.

It was noticed many times that it is easier to start an arc between metallic electrodes the readier they vaporize.

A metallic arc is more readily extinguished and harder to relight than a carbon arc. W. Duddell (The Elect. 46, p. 311, 1900) maintains an arc between copper pencils of 6 mm. diameter with a current of 3 amps. However, the arc was extinguished when put in parallel with a condensator of 0.6—5.4 mikrofard. At 0.6 mikrofard no extinction followed. When the copper pencils were replaced by cored carbons, the arc could be maintained with a current of 3 amps. in parallel with a capacity of 5.4 mikrofard.

Le Roux (Comptes Rendus Hebdomadaires de Seances de l'Academie des Sciences, Paris 65, p. 1149, 1867) succeeded to start again a carbon arc which was kept extinguished 1/25 of a second by starting the current. However, Duddell (The Elect. 46, p. 311, 1900) found that it was impossible to relight an arc between copper electrodes after it was kept extinguished more than 1/27000 of a second by starting the current. Edlund (Prog. Ann. 134, p. 250, 1868) already noticed that he could not relight an arc between silver electrodes by simply turning on the current, while he succeeded to accomplish it with ease with carbon arcs. The heat conductivity of carbon is about thirty-seven times weaker than of the poorest conductor of heat among metals. The metallic arc consists of gases cooling down and solidifying rapidly after the extinction of the arc, while carbon, whose temperature of evaporation is higher than that of metals, produces in the arc gases which keep up longer their conductivity for electricity than metallic ones. The formation of insulating oxides likewise make the relighting of the metallic arc harder than the relighting of the carbon arc. This explains the reason why it is easier to extinguish and harder to relight a metallic arc than a carbon arc.

The same explanation is applicable to metallic electrodes and their arcs at low-tension alternating currents.

At low-tension direct currents easily produce metallic arcs in most cases; however, this is not so with alternating currents. Zuchristian (Sitzungsber. d. K. Acc. d. Wiss. in Wien 102, IIa., p. 567, 1893) ascribes the impossibility to produce an arc between metals at low-tension alternating currents to the high heat conductivity of metals, owing to which property the last rapidly cool down at each half period. Zuchristian produced an unsteady arc with carbon pencils of 17 mm. in length, in-

closed in copper alloy holders. *He did not succeed at all to produce an arc between pencils of pure metals.* Sahulka (Sitz-b. d. K. Acc. d. Wiss. in Wien, 103, IIa, p. 925, 1894) confirmed Zuchristian's conclusions. Arons (Wied. Ann. 57, p. 185, 1896) tried to produce an arc between metallic electrodes, and expressed the results of his experiments as follows:

"I can affirm that it is impossible to produce an arc between two metallic electrodes by the means of an alternating current of 200 revolutions per second and at medium tension (which tension is from ten to twelve times larger than is necessary for the production of constant tension), although, when the metallic pencils were replaced by carbons of approximately the same dimensions, an arc of about 1 cm. length and high luminosity was easily produced. Blondel proved that the arc is extinguished at the end of each half period of the alternating current. The relighting is produced by a discharge emanating from the negative electrode. Arons arrived at the conclusion that it is necessary to employ high-tension alternating currents in order to produce a metallic arc.

The discontinuity of the arc between metals (as well as carbon arc) produced by a direct current seems to be established by many investigators. Lecher, for instance, proved in 1887 the discontinuity of the arc produced by a direct current between iron and platinum electrodes. Owing to the crude methods, not allowing the recording of very rapid changes in the arc, he, however, did not succeed to prove the discontinuity of the arc between silver and copper electrodes. Arons demonstrated (Wied. Ann. 58, p. 73, 1896) the discontinuity of the mercury arc.

The discontinuity of the metallic arc may, produced by a direct current, account, partially at least, for the unsteadiness of this arc on the same principles as it is thought to hinder the relighting of the alternating-current metallic arc.

Edlund (Pogg. Ann. 134, p. 250, 1868) connected silver electrodes with a galvanometer of high resistance 1/80 of a second after the arc between them was extinguished, and noticed no deflection of the needle of the galvanometer, while under exactly the same conditions carbon pencils produced a considerable deflection. Lecher (Sitzungsberichte d. Kais. Acc. d. Wiss. in Wien 95, IIa, p. 992, 1887) could not ascertain any difference of resistances between cathodes and anodes of platinum, iron, silver and copper. He found that the potential difference between a metallic electrode and any part of the gaseous arc is equal to about one-half of the potential difference between cathode and anode. As an explanation of this phenomenon he advanced the supposition that the difference of the metallic cathode and anode is not so pronounced as in the carbon cathode and anode.

(To be concluded.)

Notes on Electrochemistry and Metallurgy in Great Britain.

(From Our Special Correspondent.)

INSTITUTION OF MINING AND METALLURGY.
GOLD METALLURGY.

The Institution of Mining and Metallurgy at their meeting on Thursday, April 13, had five new papers before them for discussion. The first place was given to Mr. T. Kirke Rose's paper on *Refining Gold Bullion and Cyanide Precipitates with Oxygen Gas*. The paper was a lengthy one, the best summary being perhaps afforded by the author's conclusions.

"(1) It has now been shown that base metals can be removed from gold and silver bullion by means of a current of oxygen passed through them. (2) The losses due to volatilization and projection are insignificant, and those in the slag are moderate and can be kept low by stopping the process be-

fore the refining is complete. The initial losses in the slag, etc., including all beads, varied from 0.03 per cent of the values in Experiment No. 21, to 1.70 per cent in Experiment No. 18, when the slag was pasty, the mean being 0.73 per cent. Most of this could be brought to account at once by crushing and sieving. (3) The various impurities are equally harmless, provided that some readily oxidizable metal is left in the bullion to protect the silver. It is probable that copper, tellurium and bismuth have little action in protecting silver from oxidation. (4) Pure oxygen gas and air have the same effect, and there is little doubt that, if a deep body of metal were used, their efficiency would be, at the lowest, 80 or 90 per cent. (5) The fluxes required for fusion are borax and silica, and these are sufficient. The slag formula $2-3 (Na_2 O B_2 O_3) + 6RO-2-3 (B_2 O_3) 3SiO_2$ appears to fulfill all requirements, except that of cheapness. The cost of the fluxes in such a slag would be about $\frac{1}{2}$ d. per ounce of fine gold recovered. The proportions indicated by the formula may undergo considerable variations without any disadvantage. On the other hand, when 20 per cent of both the borax and the silica are omitted, the slag becomes somewhat pasty. The replacement of even 90 per cent of the borax by sand appears to be more than a possibility, and would greatly reduce the cost of the fluxes. The addition of iron oxide or lime may be useful if borax is omitted. (6) The gold can in great part be recovered from the slag by concentration, but the silver cannot be recovered in that way. Almost all the values can be recovered by fusion with carbon and iron, the lead and copper being reduced and carrying down the gold and silver. (7) In the treatment of zinc-box precipitate from the cyanide process, it is convenient to drive off part of the zinc by volatilization in order to save fluxes and to avoid pasty slags. (8) The metals are oxidized in succession, each in turn partially protecting the metals which are less easily oxidized than itself. Owing to mass-action, this protection is not absolute. Silver is not easily oxidized, and gold is protected by all other commonly occurring metals. (9) There is *prima facie* evidence that the cost of treatment by air is less than that of other refining processes. The relative losses are more doubtful, and cannot be determined in the laboratory. The total losses observed in the experiments, including the values left in the slag tailings, varied from 0.008d. to 2.4d. per ounce of fine gold recovered, the mean loss being 1.36d. On a large scale these should be much less."

VARIOUS PAPERS.

The second paper to be discussed dealt with *Wood Gas for Power Purposes and Gas Generators*. In it Mr. George M. Douglas described the gas-driven power house of the Moctezuma Copper Company, at Nacozari, Sonora, Mexico. (See our Vol. II, p. 104.) Coke is used on setting away the gas producers, after which wood fuel, consisting mostly of an inferior scrubby oak. An average sample of the wood gas contains 19.5 per cent H, 13.45 per cent CO, 2.48 per cent CH₄, 15.45 per cent CO₂, 48.5 per cent H₂, 0.34 per cent Cn H₃N and 0.25 per cent O. Its calorific value is 135.6 B. T. U. per cubic foot. Under normal working conditions the average is about 2.5 pounds per B. H. P. hour, and 0.11-pound coke.

The third and fourth papers may be almost described as *Prospectors Reports*. In the third, Mr. Philip Poore supplies some notes on the Prestea District, Gold Coast Colony, while Mr. R. O. Ahlers, in the fourth, deals with the new Dharwar Gold Fields. They are both partly geological, partly antiquarian, and the former is partly economic. They both form valuable contributions to the bibliography of mining districts, but do not call for other comment in these pages.

The fifth, and last, paper was purely geological, being entitled *The Cause of Border Segregation in Some Igneous Magmas*. The author, Mr. James Park, ascribes border segregation "mainly to differences of osmotic pressure in the magma, with perhaps convection currents as a contributing cause." It is suggested that if a molten magma be regarded as

a mass of rock material in solid solution, the first crop of minerals separated by cooling being inert and unable to offer ionic resistance will be carried by the transference of osmotic energy from the interior to the borders which form the coolest parts of the magma. "Manifestly this unequal struggle between different potentials of osmotic energy will continue so long as the difference of pressure exists, but a point will be reached where the pressure will be neutralized by the increasing viscosity of the magma. The minerals which first crystallize are generally basic, and these being carried outward as they form, necessarily leave the interior more silicious; or acid than the borders." Finally it is suggested that molecular concentration may be the result of differences of osmotic pressure due to unequal temperature in a homogeneous solution.

REFRACTORY AURIFEROUS SULPHIDES.

The most important of five papers which were presented at the May meeting of the Institution of Mining and Metallurgy was that by Mr. F. B. Stephens, on "The Treatment of Refractory Auriferous Sulphides at the Cassilis Mine, Victoria, Australia." The surface ore in this district is very rich, giving as much as 10 ounces to the ton, but this proportion falls rapidly with an increase in depth, so that improved methods of extraction alone permitted work to be continued in depth. The main body of ore comprises a band of dense sulphides, consisting of arsenical pyrites, iron pyrites, zinc blende, galena and carbonate of iron, whilst antimony, manganese, copper and tin occur in proportions usually of less than 1 per cent each. The gold values range from 1 to 10 ounces, averaging from 3 to 5 ounces. The plant used for treatment consists of a jaw crusher, 16 inches by 9 inches, self feeders, twenty head of 1000-pound stamps, silvered copper plates, four three-cone hydraulic classifiers, six Wilfley tables, and four Berdan grinding pans. Tailings go to settling tanks for future cyaniding, and concentrates are roasted for chlorination, whilst the galena is shipped to smelters after grinding. A drawback to chlorination in this district has always consisted in the large percentage of galena carried by most of the ores, and copper is also occasionally troublesome. Zinc blende, after roasting, is quite amenable to chlorination, our concentrates sometimes carrying 20 per cent of zinc. The galena is separated on the Wilfley tables and subjected to special treatment, thereby getting over the trouble in the furnaces due to fusion of the sulphides, a matter of some consequence with galena, running sometimes as high as 7 per cent on the ore crushed. The ore is crushed to pass a 17 by 20-mesh screen, and after crushing, the pulp is run over silver-plated copper plates, given a fall of $1\frac{1}{2}$ inches per foot, to keep the plates clear of fine galena. The amalgam yields a return of one-fifth to one-seventh of smelted gold. Before the employment of classifiers great losses of gold and mercury took place, but since their introduction the gold and mercury is caught in the galena streak on the Wilfleys. Only 20 per cent of the total gold is obtained by amalgamation. With the adoption of classifiers no occasion was found for use of Raff wheels, which were thrown out. At the end of the tables an inner sliding box was arranged in the usual concentrates box, for the purpose of catching the galena concentrates. These concentrates contained a quantity of arsenical pyrites, and were daily reconcentrated on a separate table reserved for this purpose, and three products were made. Of these, the heads containing 40 per cent of lead went to the grinding pans; the middlings, containing 15 to 20 per cent of lead, 15 to 20 per cent of arsenic, and 8 ounces of gold per ton, were shipped direct to the smelters, and the thirds went back to the ordinary concentrates for chlorination. Smelters, by the way, paid for gold, silver and lead, but deducted 1s. per unit for arsenic. The galena headings were ground in Berdan pan with mercury and a little cyanide for the period of twelve hours or less, and the ground pulp, when removed from the pans, was stirred in a tub and allowed to settle, when the top

sludge was removed, put into jute bags to drain, and shipped to smelters. Settlements at the bottom of the tub, after removal of the bulk of the mercury and amalgam, were returned to the next charge, and in this way the mercury loss was kept at a minimum, averaging for all operations at the mill from .75 to 1.05 ounce per ton crushed. The lead sludge from the grinding carried about 15 ounces gold, 40 per cent lead and 10 per cent arsenic when shipped; the gold value before grinding being sometimes as high as 60 ounces. The stamps were run at 56-100 drops per minute, with a 6-7 inch drop.

As it was usual to make from 15 to 20 per cent of concentrates, running from 3 to 4 ounces of gold, and containing over 50 per cent of the total gold value of the ore crushed, the chlorination work had to be run as cheaply as possible.

It was found that short hand-rabbed reverberatories were quite unsuited for the work, on account of the concentrates fusing too easily. As the galena could not all be separated, some form of mechanical furnace was absolutely necessary. Chlorine solutions generated from bleaching powder and sulphuric acid were used in 10-ton open wooden vats, and after comparative trials against gas, this method was retained as offering many advantages, being actually cheaper for this ore, while the extractions were the same for both.

The chlorine solutions run on had a strength of .09 to 1.2 per cent of chlorine, and from .5 to 1 per cent of sulphuric acid over and above the quantity required to combine with the bleaching powder. In this way many deleterious compounds were leached out or rendered harmless, most of the copper present being leached out in the first solution, leaving the ore in a good condition for the next solution. The first solution usually had all its chlorine used up, but on well-roasted ore the second solution ran off, showing chlorine freely. The solutions were not allowed to remain long in contact with the ore, as otherwise the chlorine became used up and the gold already dissolved was precipitated in the vat. Solutions were run on for six days, or until the off-flowing solutions failed to react well for gold. The solutions were allowed to settle in an intermediate vat before precipitation. This vat was fitted at plugs at every 6 inches for drawing off small portions for assay, and an extra check on the amount of gold due at the clean-up was obtained. Ferrous sulphate was used as a precipitant, and was made from old battery screens. A considerable amount of basic iron sulphate, etc., settled in the precipitating vats, and at clean-up time was taken into solution with sulphuric acid. Copper gave very little trouble, but it seldom went over 1 or 2 per cent in the concentrates. Owing to the use of such weak solutions not much chlorine passed into the atmosphere. Less chlorine was consumed than if dry gas had been used on this ore. An extraction of about 85 per cent was obtained on well-roasted ore, the loss amounting to 1 to 2 dwt. per ton crushed. Shipping to smelters and treatment charges amounted to £6 per ton, whilst the costs, plus the value of gold left in, did not amount to half that. The drawback to the use of 10-ton vats was the fact that a little carelessness on the part of a furnaceman spoils the extraction of a whole vat.

Cyanide works were erected to treat the accumulated tailings, but much difficulty arose in leaching these owing to the large percentage of talc and mica; further, the slimes running high in concentrates gave rise to clogging, and the solutions could not be allowed to remain in contact with the sands. Solutions which ran off quite clear, on standing became quite milky with magnesia hydrate, and this precipitate settled thickly on the zinc and spoiled its efficiency. Extractions ranged between 60 and 65 per cent, and the consumption of cyanide kept at the moderate amount of 75 pounds. A treatment of seventy-two hours was given, but a longer treatment gave no better extractions. The use of an oxidizing agent in experiments increased the extractions 10 per cent. The water supply destroyed permanganate readily; organic matter in the water may

have accounted for this, as the water came from peat bogs. If the ore were leached with distilled water a heavy consumption of permanganate followed on the magnesium sulphate, which seems rather strange.

For assaying the cyanide solutions an electrolytic method is employed. For a weekly output of 270 tons of ore milled and mined, 54 tons of concentrates are chlorinated, 300 tons tailings are cyanided, and four tons of galena are shipped for smelting. The cost per ton of ore crushed is stated to be £18.0 (excluding directors' fees and Melbourne office expenses). When it is remembered that 27 to 30 per cent bleaching powder cost £18 per ton, chamber sulphuric acid £11 per ton, and wood—the only fuel available—cost from 14s. to 18s. per cord of 125 cubic feet, and that explosives varied from £3.26 to £4.26 per case, it will be realized that a cost of \$7 per ton of ore mined is very satisfactory.

Of the remaining papers none calls for abstracting, although each was of interest. Mr. F. J. Stephens' "Notes on a Low-Grade Copper Ore Deposit in the Himalayas," does not betoken any prospect of the immediate cheapening of this metal. Containing only 2 per cent of copper, poor transport facilities are likely to defer its development, "notwithstanding the undoubted mining facilities which exist, such as abundant and cheap timber, and some of the cheapest labor in the world." Mr. H. Erelich's "Chinese Methods of Mining Quicksilver" is an account of what we should call in England a threatened and decaying industry once worked on a much larger scale. Yet threatened industries—like threatened men—live long, and Mr. Erelich anticipates the maintaining of a constant output for many years to come. Much of what the author says respecting the peculiar habits of the Chinese miner would, if the locale were the Transvaal, make an admirable anti-Chinese labor pamphlet. Mr. A. N. Mackay, in his paper, "Precipitation of Gold from Cyanide Solutions and Absorption of Gold by a Wooden Leaching Vat," deals with two different subjects, the one due to the use of too alkaline a solution with a too slow percolation rate, the latter showing that the red pinewood of a leaching vat absorbed gold to the extent of 12 ounces 8 dwts. per ton of wood. Lastly, Mr. H. R. Sleeman's account of an Egyptian gold mine describes some abandoned ancient workings now being dealt with by modern methods.

THE ROYAL SOCIETY'S CONVERSAZIONE.

The Royal Society's *Conversazione* held on May 17 was chiefly interesting by reason of the exhibits of scientific value which excited much attention. On the metallurgical side, mention must be made of a series of microscopic slides and photographs illustrative of an ingenious method of detecting the presence of phosphorus in steel, exhibited by Mr. J. E. Stead, who also showed an apparatus for polishing and preparing metals for microscopic examination.

Dr. Glazebrook showed some high temperature furnaces constructed of rare earths, such as are used in Nernst lamps. They are available for temperatures between 800° C. and 2000° C., and one had recently been used to determine the melting point of platinum.

Mr. W. Rosenhain showed some transverse sections of slip bands and other microscopic features of metallic surfaces. The surfaces to be examined in section are first coated with a thick electrolytic deposit of copper or other metal, and a section is then cut through the compound specimen. The line of contact between the original metal and the electro-deposit is perfectly sharp under the highest magnification, and reveals what is claimed to be an accurate sectional view of the original surface.

Mr. R. S. Hutton exhibited new models of laboratory electric furnaces which were interesting. The furnaces consisted of carbon tubes or rods heated by the electric current. The method employed for conveying the current to the carbon by soldering water-jacketed sleeves to the electro-coppered ends of the carbon formed a novel feature of the construction. As

will be remembered, this tube type of furnace formed the subject of a paper read before the Faraday Society.

THE FARADAY SOCIETY.

First among London scientific societies to start their session in October last, the Faraday Society has been the last to break up for a vacation. While other societies have held their conversaciones in their own halls, as the Institutions of Civil and Mechanical Engineers, or have gathered around cases of stuffed birds and the vertebra of the *Diplodocus Carnegii* at South Kensington, as did the Institution of Electrical Engineers, or have held a garden party in the Royal Botanical Gardens, like the Society of Arts, the members of the Faraday, eschewing the formal pursuit of pleasure, broke all their own records with a four-paper night on July 3 last. The following was the order of procedure: Mr. Sherard Cowper-Coles, "Some Notes on the Rapid Electrodeposition of Copper." 2. Prof. W. W. Haldane Gee, "The Use of Balanced Electrodes." 3. Messrs. H. D. Law and F. Mollwo Perkin, (a) "Electrolytic Oxidation of Hydrocarbons of the Benzene Series." Part II., "Ethyl Benzene, Cumene and Cymene"; (b) "Electrolytic Analysis of Antimony." 4. Messrs. R. S. Hutton and J. R. Beard, "Notes on Heat Insulation, Particularly with Regard to Materials Used in Furnace Construction." Nominally, a four-paper night, it will be observed that the two sections of Mr. Mollwo Perkin's and H. D. Law's papers are on entirely different subjects. There

was a good attendance of members and the papers were well discussed. As to what was said, I propose to defer my notes until next month. Although societies may temporarily cease from meeting, the publication of technical journals and the duties of their correspondents do not cease. I, therefore, reserve my notes and comments for the time when certain of my sources of news and matter run dry.

MARKET QUOTATIONS.

Nothing particular has to be reported with regard to the prices for chemical substances, save that copper sulphate has receded to £19.10.0 per ton, and shellac is again cheaper, at £7.5.0 to £7.7.0 per cwt.

After the wild gambling of May, Cleveland pig iron has steadied down to £25.6 per ton, but the outlook in the iron and steel industries being promising, a rise is expected. Copper was fairly quiet, very little speculative buying being indulged in. The closing price was £66 per ton. Tin has risen to £139.12.6 per ton, on account of lower shipments from the Straits. Aluminium ingots are listed at £130 per ton, aluminium wire and sheet being quoted respectively at £168 and £166 per ton. English lead has been fairly steady, improving to £14 per ton towards the end of the month for ingots, and to about £15 for sheets. Quicksilver is unchanged, but zinc has arisen £1 to £27.7.6 per ton. I have also to chronicle a rise in the price of platinum to £4.4.0 per ounce.

ANALYSIS OF CURRENT ELECTROCHEMICAL PATENTS.

By GEORGE P. SCHOLL, PH. D.

ELECTRIC FURNACES AND FURNACE PRODUCTS.

Process of Treating Phosphate Rock.—F. J. Machalske, Brooklyn. Patent 789,438, May 9, 1905. Application filed Feb. 25, 1905.

The process comprises the crushing of rock, containing tricalcium phosphate, especially low rock, with less than 70 and down to 50 per cent of phosphate, and the subsequent mixing of the crushed product with an alkali metal chloride of carbon, after which the mixture is heated to a temperature sufficient to decompose the rock and to effect the production of phosphoric chloride and calcium and sodium carbides. The preferred mixture consists of the crushed rock, sodium chloride and an excess of carbon in the form of broken coke. The smelting is preferably effected in an electric furnace; the charge should be free from water, and air is excluded from the furnace. The heating may be done in an electric furnace, either of the arc or the resistance type. A current, preferably alternating, of from 2000 amps at 50 volts to 3500 amps, at 40 volts, is stated to be sufficient for a furnace of the usual size. The phosphoric chloride is led off from the furnace, and may be treated with water for the production of phosphoric and hydrochloric acids. The calcium-sodium carbonate may be treated with water for the production of acetylene, calcium hydroxide and sodium hydroxide.

Process of Treating Phosphate Rock and Producing Compounds of Phosphorus and Nitrogen.—F. J. Machalske, Brooklyn. Patent 789,439, May 9, 1905. Application filed Feb. 25, 1905.

The initial step of the process consists in the crushing of the low-grade material as above and raising it with alkaline chloride and carbon. The mixture is then smelted at a temperature sufficient to decompose the rock and to effect the production of phosphoric chloride, the smelting being preferably carried out in an electric furnace. Nitrogen or air, or another gas containing nitrogen, is then blown through or brought in

contact with the charge in the melting furnace, and the calcium and sodium combine with the carbon and nitrogen to produce cyanamides. The production of the cyanamides is stated to be presumably effected in two stages, the first being the production of the carbides, which then, in the second stage, combine with the nitrogen. The phosphoric chloride is treated as above, while the calcium-sodium cyanamides may be treated with water for the production of calcium and sodium carbonates and ammonia. The cyanamides may also be mixed with sodium carbonate and smelted in an electric furnace for the production of sodium cyanide.

Process of Treating Phosphate Rock and Producing Phosphorus Chlorides and Alkali Metal Cyanides.—F. J. Machalske, Brooklyn. Patent 789,440, May 9, 1905. Application filed Feb. 25, 1905.

The process comprises the same steps as the preceding one, resulting in the formation of calcium-sodium cyanamides. The latter are smelted in an electric furnace, either of the arc or the resistance type, with sodium carbonate. An alternating current is preferable, of from 800 to 1000 amps, at 50 volts. The sodium carbonate is preferably produced by treating another portion of the calcium-sodium cyanamide with water.

ELECTROLYTIC PRODUCTION OF METALS AND COMPOUNDS.

Centrifugal Apparatus for Electrolytic Purposes.—R. V. Hauser, Erie, Pa. Patent 790,055, May 16, 1905. Application filed Sept. 17, 1904.

The apparatus is to be used particularly for the manufacture of persulphate. It consists essentially of a revoluble drum, provided with an anode of lead, corrugated and perforated. The anode is parallel to the outside wall of the drum, a space being, however, left between them. The cathode is preferably in the form of a cylinder provided with holes. Sulphuric acid is supplied by a pipe into the space enclosed by the cylindrical cathode, while an aqueous solution of ferrous sulphate is fed

into the space outside the cathode cylinder. These liquids are claimed not to mix, on account of their different densities. The electric current causes the liberation of SO_2 in its nascent state at the anode, and transforms persulphate of iron, $\text{Fe}_2(\text{SO}_4)_3$. This substance has a comparatively high specific gravity, and when the apparatus is rotated the centrifugal force tends to force it outwards through the holes, and it eventually finds its way to the discharge pipe. In the course of action the anode becomes coated with peroxide of lead, which latter, it is claimed, improves the action.

Electrolytic Apparatus.—F. A. Decker, Philadelphia. Patent 789,721, May 16, 1905. Application filed Feb. 29, 1904.

The invention is stated to relate to electrolytic apparatus designed for the recovery of zinc, magnesium or other apparatus from acid or neutral solutions, such as spent battery fluids and solutions obtained in the reduction of metal from its ores or scrap. The apparatus is divided into two compartments by means of a porous diaphragm. The positive electrode of the cell may be of carbon, peroxide of lead, or other suitable material. The negative electrode is formed by a series of trays of the usual material, which trays are arranged one above the other. A tube extends from the bottom to the top of each tray, thus providing a passage through the trays for the better circulation of the gas and the fluid contained in the compartment.

Apparatus for Obtaining Oxides of Alkali Metals.—C. W. Roepper, Philadelphia, and W. E. Harmon, Mechanic Falls, Me. Patent 790,922, May 30, 1905. Application filed July 22, 1903.

The apparatus is of the so-called mercury type, the circulation of the mercury being effected by a wheel. The patentees have found that the so-called "discharging surface," that is, the place where both the amalgam and water are in joint contact with a conducting surface, and where, therefore, the alkaline element, which has been taken up by the mercury, shows the greatest tendency to disassociate itself from the amalgam, is most usefully provided in the wheel itself. The latter, or at least that part of it which comes into contact with the amalgam, is therefore constructed of iron or nickel or carbon, and in such a manner as to increase the extent of the contact of the amalgam with the discharging surfaces. The slots, or pockets, in the wheel are, therefore, more or less filled with granules of electrically conductive material such as lumps of iron or carbon. The periphery of the wheel is surrounded by wire netting, so as to keep this material in the pockets. One of the forms shows the pockets sub-divided into numerous smaller compartments by wire plates or perforated metal plates forming circumferential flanges. The actual material of the body of the wheel is comparatively immaterial, provided the surfaces which come in joint contact with the water and amalgam are made of electrically-conductive material.

Process of Manufacturing Ammonia.—W. Hoops, Pittsburg. Patent 791,194, May 30, 1905. Application filed Dec. 14, 1904.

The patentee has discovered that when aqueous solutions of ammonium sulphate and another salt are electrolyzed in respectively opposite sides of a porous diaphragm, in a cell having a cathode of carbon, iron or another insoluble material, ammonia is set free at the cathode and sulphuric acid at the anode, and that the compound formed by the liberated acid may then be utilized. The operation is carried out in a two-compartment cell, one containing an iron cathode and the other a carbon anode. The cathode compartment is filled with a solution of ammonium sulphate, and the anode compartment with a solution of common salt. The whole may be heated to about 100°C . The ammonia, in whole or in part, is stated to go off from the cathode compartment with the free hydrogen, from which it may be dissolved by passing it through water, in which it dissolves. The hydrogen is collected and

may be used for producing hydrochloric acid with the chlorine liberated at the anode. Ammonium sulphate and sodium chloride are added to the bath from time to time, to replace that which is lost. No further data concerning particulars of electrolysis, current density, etc., are given.

Electrode.—Mataichi, Yasuda, Tokio, Japan. Patent 791,308, May 30, 1905. Application filed Jan. 21, 1905.

The construction provides for carrying off the hydrogen bubbles generated in the electrolysis of alkaline chloride, being so arranged as to permit the entrance of the electrolyte but not the escape of bubbles. The electrode consists of an outer casing of porcelain, glass or other insulating material. On one side is a partition, also of insulating material, forming a passage. The electrodes consist of two plates, one of the plates being rigidly secured to the inner face of the wall, and the other member of the electrode is preferably disposed within the space secured by an offset. The action of the hydrogen bubbles in carrying the solution with them effects a circulation of the electrolyte.

Electrolytic Deposition of Metals.—N. C. Harrison, London, England, and J. Day, Westonsuper Mare, England. Patent 791,341, May 30, 1905. Application filed July 19, 1901.

The invention provides for the use of concentrated solutions and high-current densities in the deposition of metals. This is accomplished by renewing the solution in contact with the cathodes continuously. For carrying out the invention a highly polished cathode of circular cross-section is used, *f. i.* a cylindrical or conical cathode, mounted vertically in the depositing vat and free to revolve on its own axis. The layer of electrolyte contiguous to the cathode is constantly renewed by means of jets of electrolyte, which are directed tangentially against the cathode. As shown in Fig. 1 the apparatus contains a cathode cylinder *x*, located in vat *a*. A series of vertical pipes *e e* are disposed equidistant from each other and concentrically around the cathode *x*. These vertical pipes are closed at their upper ends, but provided with holes at frequent intervals, in such a manner that a tangential stream will be directed on the surface of the cathode. It has been found in practice that copper can be successfully deposited on a cathode of about 3 inches diameter, by employing vertical pipes located at or about $2\frac{1}{2}$ inches from the center of the cathode, and spaced apart about $2\frac{1}{2}$ inches from each other concentrically around the cathode. The holes were $3\text{--}32$ inch, and were bored at intervals of $\frac{1}{2}$ inch.

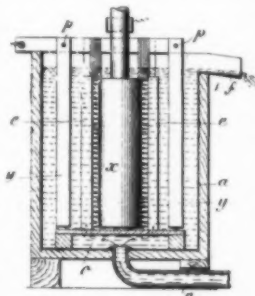


FIG. 1.—DEPOSITION OF METALS.

Process of Extracting Zinc from Its Ores.—A. G. Betts, Troy, N. Y. Patent 791,401, May 30, 1905. Application filed Feb. 20, 1905.

The process consists in first bringing the zinc into solution, for which any of the usual methods may be used, the inventor, however, preferring to use sulphuric acid in lixiviating, thus producing a solution of zinc sulphate. This is electrolyzed with an insoluble anode and with a liquid metal cathode, preferably of mercury or mercury containing zinc. Oxygen is produced at the anode and sulphuric acid remains in solution. Practically all the zinc is removed and the solution of sulphuric acid thus obtained may be used for lixiviating, thus producing a solution of zinc sulphate. This is electrolyzed with an insoluble anode and with a liquid metal cathode, preferably of mercury or mercury containing zinc. Oxygen is produced at the anode and sulphuric acid remains in solution. Practically all the zinc is removed, and the solution of sulphuric acid thus

obtained may be used for lixiviating further amounts of ore. The zinc mercury alloy, which preferably contains less than 5 per cent of zinc, is then removed to another electrolytic cell, containing a solution of a zinc salt, preferably one of zinc chloride. Into this solution a zinc cathode is introduced, while the mercury-zinc alloy is connected as anode. An electric current is then passed, whereby the zinc is dissolved from the anode and deposited on the cathode. The mercury, after the removal of a large part of the zinc, is returned, to go through the same process over again.

Electroplating Apparatus.—C. T. Pratt, Frankfort, N. Y. Patent 790,747, May 23, 1905. Application filed Oct. 3, 1903.

The apparatus is intended for plating the inside of ferrules, rings, bands, slip nipples, etc. The articles are strung on the anode bar, which is then inserted in the galvanizing solution. They rest upon a series of V-shaped troughs, along the inner walls of which extend one or more metallic strips. The latter form the connection with the negative terminal of the current, and thus constitute the articles to be plated cathodes. When the current is passed the inside of the hollow articles will be covered with a deposit.

Process of Stripping Tin.—D. W. Hemingway, Walthamstow, England. Patent 791,555, June 6, 1905. Application filed Oct. 29, 1902.

The process depends upon the use of a solution of persulphate of iron, which is contained in the tank A in Fig. 2. The bath is acidified by the addition of sulphuric acid. The tinned iron clippings from which the tin is to be removed are immersed in this solution. After the tin has become dissolved a valve *a* in the pipe *a'* is opened and the solution drawn into

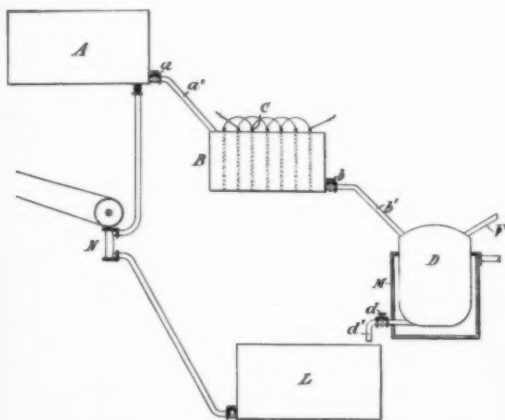


FIG. 2.—STRIPPING TIN.

the electrolytic vessel B, where it is electrolyzed between iron anodes and copper or tinned copper cathodes. When the greater portion of the tin has been deposited upon the cathode, the electrolyte passes by way of the valve *b* and pipe *b'* into the vessel D. In this vessel the solution is restored by the addition of nitrate of soda or potash, either alone or with sulphuric acid. Other oxidizing agents may be substituted for this mixture. The vessel D has an outlet F for the nitrous vapors, and is surrounded by a jacket M, for the purpose of heating its contents. After the electrolyte has become sufficiently oxidized, the liquor is withdrawn through valve *d* and pipe *d'* into the vessel L, where pump N transfers it back into the vessel A. When the quantity of persulphate of iron employed is to be reduced to a minimum, the tin scrap may be added directly to a solution of nitrate of soda, or potash, mixed with sulphuric acid, from which the tin is then recovered by electrolytic or other means.

Method of Making White Lead.—C. P. Townsend, Washington, D. C. Patent 791,956, June 6, 1905. Application filed March 22, 1904.

The process is carried out in an apparatus divided into two compartments by a suitable diaphragm. As shown in Fig. 3 it consists of a wooden vessel lined with sheet lead, the lining being treated with a solution of paraffine. The diaphragm 2

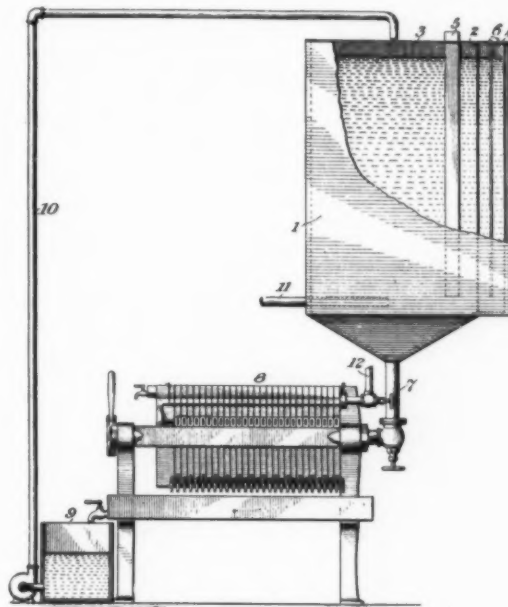


FIG. 3.—PRODUCTION OF WHITE LEAD.

is intended to prevent the compounds of lead present in the solution contained in the anode compartment, the so-called anolyte, from coming into contact with the cathode. Vegetable parchment, or parchment paper, is particularly suited for this purpose, and it is believed to be essential that the membrane should be substantially impermeable as a filter, as no useful results could be obtained by the use of tissues or woven fabrics. The diaphragm divides the vessel into two unequal compartments, namely, a relatively large anode compartment 3, and a relatively small cathode compartment 4. The anode 5 is composed of lead, while the cathode 6 may be a sheet or mesh of any suitable material. The white lead is formed in the anode compartment and is continually drawn off by a pipe 7 to a filter press 8 or another suitable separator. The filtered electrolyte passes into a tank 9, whence it returns to the electrolytic vessel by means of a pump and pipe 10. Carbon dioxide is introduced either directly by a pipe 11 into the anodic solution, or it may be introduced at any point in its circuit. As the cathodic solution apparently does not take part in the reaction by which the pigment is produced, the cathode compartment may be very small as compared with the anode compartment, or may be such that the diaphragm is in substantial contact with the face of the cathode. A pipe 12 serves for washing the pigment collected in the filter press, and a sufficient amount of the first washings should be added to the electrolyte in order to maintain its original volume. The process is not limited to any particular electrolyte, the essential requirement being that it should comprise a mixture of salts in solution, one of them being capable of yielding a solvent for lead at the anode, while the other must necessarily be a soluble carbonate or bicarbonate. As a solvent salt, preferably one of the soluble acetates, such as sodium acetate, is used. The cathodic solution may have the same initial composition as the anodic, but may initially consist of water or of any solution not incompatible with the operation.

Process of Electrodepositing Antimony.—A. G. Betts, Troy, N. Y. Patent 792,307, June 13, 1905. Application filed May 20, 1904.

The process, evidently intended to be used in the refining of electrolytic slimes, as described by Mr. Betts in *ELECTROCHEMICAL AND METALLURGICAL INDUSTRY*, Vol. III., p. 141, depends upon the use of an electrolyte consisting of antimony trifluoride. As this compound is not decomposed by water into basic antimony compounds and free acid, a solution can be prepared which is only slightly acid. In addition, there are preferably metallic salts present, which will not deposit on the cathode with the antimony, and alkali sulphates and fluorides are found to be most suitable for this purpose. In case an antimony anode is being used, alkali chlorides may be employed. The presence of sulphuric acid in the electrolyte is stated to be desirable, a suitable solution containing 8 per cent SbF_3 , 2 per cent Na_2SO_4 , and 4 per cent H_2SO_4 . A current density of 10 amps. per square foot of anode surface is stated to give a good deposit of antimony when the solution is pure, i. e., free from metals which would deposit with the antimony. By electrolyzing such a solution, with an insoluble anode of lead, oxygen escapes at the anode, leaving a solution containing hydrofluoric acid. The anode should consist of lead rods, surrounded by a layer of porous material, which is intended to prevent the tendency of the antimony to become oxidized to the pentavalent form at the anode, in which form it has a dissolving action on the metal at the cathode, where it is again introduced to the trivalent compound. Such a layer of porous material consists preferably of a few layers of cloth surrounding each anode rod. If the cloth is absent, resolution is stated to occur to such an extent that the actual yield of metal deposited at the cathode is reduced to 60 per cent of the theoretical, while the yield rises to from 90 to 95 per cent with anodes wrapped in cloth. The area of the anode should not be over half that of the cathode, inasmuch as with the higher anode, current density to formation of pentavalent antimony compounds is reduced. The voltage required, when insoluble anodes are used, is given as usually from 2.5 to 3 volts. As hydrofluoric acid is a good solvent for antimony oxides and basic compounds, the solution, after electrolysis, is in suitable condition for dissolving fresh amounts of basic antimony compounds from which it is desired to prepare metallic antimony.

Process of Treating the Metal Mixtures Produced as a By-Product in Electrolytic Metal Refining.—A. G. Betts, Troy, N. Y. Patent 793,039, June 20, 1905. Application filed May 20, 1904.

The process provides for the treatment of anode slimes, usually containing free or combined lead, copper, arsenic, antimony, silver and gold, with less important quantities of other elements, as applied to the treatment of such slimes, from the electrolytic refining of lead, which consist mostly of metallic lead, copper, arsenic, antimony and silver, it comprises as the first step the agitation of the slime, in a lead-lined tank, with a hot solution of ferric sulphate containing, say, 2 per cent of free sulphuric acid and 4 per cent of iron as ferric sulphate, an excess of the latter salt being present above that required to carry out the reactions. During this operation the copper goes into solution as copper sulphate, and the iron is reduced to the ferrous condition, while lead forms PbSO_4 , arsenic $\text{As}_2\text{O}_3 + \text{free H}_2\text{SO}_4$, antimony $\text{Sb}_2\text{O}_3 + \text{free H}_2\text{SO}_4$, and bismuth $\text{Bi}_2\text{O}_3 \cdot \text{H}_2\text{SO}_4 + \text{free H}_2\text{SO}_4$. The mixture is then allowed to settle, and the solution drawn off from the residue, which latter is washed by decantation with hot water. The wash water is run over copper so as to precipitate the silver, after which it passes over iron in order to deposit the copper, and is then run to waste. The solution drawn off from the residue is agitated several hours in contact with metallic copper in order to precipitate the silver. The free acid formed during solution of the slimes is neutralized by the addition of copper oxide to the solution. Roasted matte may be used instead, but this is added to the

solution when the iron is in the ferric state, that is, before it has been reduced by its action on slime, inasmuch as its acidity is then less and less iron is dissolved from the matte, while the ferric sulphate assists in the solution of any cuprous oxide present. The acidity of the solution is advantageously as low as possible when roasted matte is used. The solution, after treatment to precipitate the silver as described above, is first passed through the cathode compartments of an electrolytic cell, which is provided with copper cathodes, on which latter it deposits the copper which it contains. It then passes through the anode compartments of the cell, by which operation the ferrous sulphate is oxidized again to ferric sulphate and the solution thus regenerated. The arsenic of the slime accumulates in the solution and is kept from reaching too high a concentration by either withdrawing a part of the solution at a convenient stage of the process and cooling it, in order to crystallize out the arsenious acid, or by removing it by chemical methods, or by an occasional purification of the entire solution. The process of reoxidation can also be carried out by other means than electrolysis, for instance, by the use of nitric acid. In this case the too great accumulation of copper and arsenic are prevented by crystallizing them out as copper salt and arsenious acid. Nor is there any limitation as to the use of iron as the carrier of oxygen from the oxidizing tank to the slimes-treatment tank, inasmuch as chromium or manganese compounds in the form of permanganic or chromic acids may be used. They may be reduced by the action on the metals of the slime to manganous salts and trivalent chromium salts, which can be reoxidized in various ways, as, for instance, by electrolysis or reaction with peroxide of lead. Persulphates, produced electrolytically, may also be used as oxidizing agents. The reoxidation of a part of the reduced compound and the reduction of another part from its oxidized condition by action on the metals of the slime in the same solution may be performed as follows: A solution may be used of an acid forming a readily soluble salt of lead, such as fluosilicic acid, in the presence of compounds of metals like iron, chromium or manganese. Ferric salt is reduced to ferrous salt, chromic acid to chromic salt, or permanganic acid to manganous salt by the action on the metals of the slime. At the same time the ferrous salt present is being oxidized to ferric salt, the chromic salt to chromic acid, or the manganous salt to permanganic acid by reaction with the acid in solution and lead peroxide which is added. Only a small amount of the oxygen carrier is stated to be present. The resulting solution of lead salts and salts of metals of the slime can be treated for precipitation of the silver by metallic copper, of the copper, arsenic, antimony, etc., by precipitation with metallic lead. The resulting solution of lead salt can then be electrolyzed with a lead cathode for the deposition of lead, and with an insoluble anode of carbon for the deposition of peroxide of lead. The latter and the acid set free during electrolysis can be used for the treatment of a fresh quantity of slime. When using electrolysis, in order to reoxidize the compound reduced in attacking the slime, the employment of a sulphate solution is not absolutely necessary, inasmuch as there may be electrolyzed a solution of iron and lead fluosilicates, or iron and lead salts of other acids forming readily soluble lead and iron salts, depositing lead on a cathode and oxidizing ferrous to ferric salt at the anode, which ferric salt is then applied to the slime for the solution of silver, copper, arsenic, bismuth, etc. From the solution silver can be precipitated by metallic copper, and the other metals dissolved as well as the copper brought into the solution by the precipitation of the silver, can be precipitated with metallic lead, after which the solution is brought back to the condition of containing iron and lead fluosilicates. It is also not absolutely necessary to use hydrofluoric acid for dissolving the antimony from the insoluble residue left after the extraction of the copper, etc. Other suitable solvents are hydrochloric acid and cream of tartar solution. Only such oxidizing reagents are used for carrying

out the process, such as ferric sulphate, *f. i.*, which will not dissolve gold, even if the oxidizing agent is present in excess.

Electrolytic Cell.—J. F. Kelly, Pittsfield, Mass. Patent 792,597, June 20, 1905. Application filed Sept. 9, 1904.

The cell is particularly intended for the electrolysis of alkali sulphate, and is divided by diaphragms into three compartments. The electrolyte is contained in the middle compartment, and one of the other compartments contains the anode and the other one the cathode. There are thus no electrodes in the central compartment which contains the electrolyte. The anode is of lead, while the cathode is of iron. The liquid, which receives the products of the decomposition of the electrolyte is water, and fills the anode as well as the cathode compartments. The diaphragms are composed of one or more sulphates of the alkaline earths, calcium sulphate being chosen on account of its cheapness, though barium sulphate may be used with equal effectiveness, or a combination of the two. They are thin plates, secured suitably in the cell. The electrolyte is a saturated solution of sodium sulphate, and is maintained concentrated by placing crystals of the salt at the bottom of the central compartment. When the liquid in the anode and cathode compartments has become sufficiently dense it is withdrawn at a fixed rate.

STORAGE BATTERIES.

Storage Battery.—R. Macrae, Chicago. Patent 791,350, May 30, 1905. Application filed Aug. 4, 1904.

The battery is so constructed that the container is formed with double walls, the interior of the container holding one active element, the other active element being without the container. The latter, between its walls, is filled with acid or another electrolyte. The container is made of wood, so as to get the advantage of its yielding qualities for use in automobiles.

Relief Valve for Storage Batteries.—A. P. Perry, Chicago. Patent 789,877, May 16, 1905. Application filed May 22, 1903.

The relief valve is applied to the top of the cell, and consists of a cap, provided with a central opening and an internally threaded lower portion. Within the latter is placed a rubber diaphragm, which has a microscopic central opening. Access to the interior of the cell can be readily obtained if desired.

GALVANIC ELEMENTS.

Battery.—W. C. Banks, New York. Patent 790,866, May 23, 1905. Original application filed April 11, 1901. Divided and refiled Jan. 5, 1905.

The negative element of the cell is composed of a metallic basket with perforated walls, the bottom of which is formed of a cone-shaped piece of perforated tin, the smaller end of the cone being upwards. It is usually filled to within an inch of the top with black oxide of copper.

Battery Solution.—F. M. Holmes, Marionville, Mo. Patent 790,502, May 23, 1905. Application filed Jan. 25, 1904.

The solution is produced by dissolving a suitable gum in water and mixing it with the acid-exerting fluid of the battery. The gum is stated to form a coating on top of the solution, and to retard evaporation. The solution is claimed to work longer and to be more satisfactory than the old solutions.

Battery.—W. C. Banks, New York. Patent 787,716, April 18, 1905. Application filed Oct. 22, 1904.

The battery is so constructed that it may easily be taken apart and cleaned. It consists essentially of a zinc cylinder, upon which rests an insulating plate, which latter, in its turn, serves as support for a perforated cup, which contains the depolarizer.

Battery.—W. C. Banks, New York. Patent 787,715, April 18, 1905. Application filed Oct. 20, 1904.

This construction is similar to the above, of the zinc-copper oxide type; the zinc electrode, of tubular form resting upon a porcelain support, while it in turn carries the cylinder, usually of perforated tin, which contains the depolarizer. A rubber gasket, placed over the top of the zinc, insulates the latter from the tin depolarizer-basket.

MISCELLANEOUS.

Medical Ozonizer.—M. Otto, Paris, Assignor to American Ozone Co., Niagara Falls. Patent 790,655, May 23, 1905. Application filed June 6, 1903.

The ozonizer consists of a glass tube in which the electrodes are mounted, namely, one electrode inside the tube and the other covering the outside wall of the tube. The tube is provided at its lower extremity with an air-injecting tube 3, by which the air to be ozonized is injected into the chamber, passes between the electrodes and passes out through a trumpet-shaped opening, where it is inhaled by the patient.

Electrical Ozonizer.—C. A. Sahlstroem, Ottawa, Can. Patent 788,557, May 2, 1905. Application filed June 21, 1904.

The positive electrodes in this apparatus are constructed out of a sheet of metal having triangular-shaped tongues stamped out of it. The latter are bent so that the points are in contact with an insulating sheet placed on top of the metallic sheet. The negative electrode is made of wire gauze. The ozonizer is built of successive layers of an insulating plate of glass, a wire gauze electrode, another glass plate, and a positive electrode, with the points in a direction towards the negative plate, this succession being repeated as often as desired. When a sufficiently high voltage is used between the terminals the current passes through the glass plates from the points of the positive electrode to the nearest point where the negative electrode is in contact with the plate. Pressure is exerted upon the end plates, so as to ensure good contact of the respective electrodes with the glass plates. An alternative form shown has circular electrodes, placed around glass tubes, but the principle is the same as in the first one. By means of an electrically-driven fan air is forced through the apparatus.

SYNOPSIS OF PERIODICAL LITERATURE.

A Summary of Articles Appearing in American and Foreign Periodicals.

INDUSTRIAL ELECTROCHEMISTRY.

Electric Steel Manufacture.—We recently noticed in these columns Prof. W. Borchers' paper presented before the Association of German Iron Metallurgists on the present status of iron and steel manufacturing in the electric furnace. The discussion which followed the paper is given in full in *Stahl und Eisen*, June 15. G. Gin remarked that he has given the

license for using his process in Germany to W. Brueninghaus, who has formed the company Deutsche Elektrische Stahlwerke, with the coöperation of Siemens & Halske and two other German firms. This company has installed an experimental plant at Plettenberg, where the furnace is in operation, and Gin hopes to give shortly data of the exact cost. He considers electricity as a "luxury force," that is, one which should

be used only for such special purposes where it is distinctly superior to others. To explain the high heat efficiency of electric furnaces he points out that in his furnace a space of 1 cubic decimeter contains a mass of 7 kg, and stores about 2700 calories from 0 to 1800°, while the gases of the open-hearth furnace within the same volume weigh only 2 decigrams, and can store only one-quarter of the above calories. This shows that operations in an electric furnace take place in volumes of small dimensions, and that, therefore, the losses due to heat conduction and radiation are smaller than, for instance, in the open-hearth furnace. On the other hand, operations in the open-hearth furnace are difficult, on account of the reactions of the gases on the metal bath, and it is very difficult to maintain at will a neutral or reducing atmosphere. In the electric furnace the atmosphere is as desired, and its action may be considered as of no account. He claims that with the electric furnace it is possible to make steels of almost mathematically exact composition and properties. He does not think that the electric furnace will revolutionize the metallurgy of iron, but that it will mainly be used for refining purposes. He does not believe that our present steel makers have a right to say that their steel needs no refining. The electric furnace will not be the competitor but the co-worker of the present metallurgical furnaces, and its introduction into practice depends on the utilization of the blast furnace gases. Prof. H. Wedding referred to the remark that the electric refining of steel is a luxury; nevertheless, it is possible to see a progress in this direction. He thinks, however, it is useless to make further experiments on the use of the electric furnace for the reduction of iron from ores. For the reduction of iron oxides which cannot be accomplished by dissociation, carbon monoxide should be used, not carbon. By means of carbon monoxide reduction is obtained at low temperatures (up to 700° C.) without considerable loss of heat, but since in the electric arc furnace the temperature is very high the reduction will be accomplished by carbon, and this should be avoided. (He did not mention resistance furnaces.) Eichhoff pointed out that for the interests of the army and navy, for automobile manufacture, etc., better qualities of steel are required. He then spoke at some length on the Heroult process, by which about 4000 tons of steel have been made of very best quality. He pointed out that the quality of the product depends intimately on the slag. The more energetic the action of the slag on the bath the more quickly will the deleterious constituents of the bath be removed. In the Heroult furnace the slag has a very high temperature, and its action is very energetic. This explains why by this process steels have been made with properties different from those kinds made at present. "The commercial manufacture has shown that these new properties represent an improvement, that the properties of the steel are bettered, since with the same content of carbon the steel is tougher. For instance, Heroult steel with 0.75 per cent carbon, is as tough as ordinary steels of 0.5 per cent. Another interesting property of electric steel is that it has a high limit of elongation." * * * "The hope of Heroult that he will be able to work out his process so that its cost will be almost the same as that of Bessemer and open-hearth steel, should be considered interesting, but it must first be proven correct in the practice." Engelhardt spoke of the Kjellin process, and thought that a disadvantage of the Heroult process is that there exists at least a possibility of particles of the carbon electrodes dropping off and spoiling the bath.

Electrochemistry and Engineering.—Prof. F. Haber, well known to American electrochemists from his visit to this country some years ago, recently delivered a suggestive address before the Munich local section of the Association of German Engineers, which is printed in *Zeit. f. Elektrochemie*, April 28. He first discussed the production of electrical energy by means of a carbon cell, explaining the action of the Jacques cell in accordance with the theory given in his St.

Louis Congress paper (our Vol. II., p. 401). He thinks that the solution of the problem at a temperature below 400° C. is not impossible, especially since the reaction of carbon monoxide with atmospheric oxygen begins to get very slow only at about 300°; however, nothing of a definite nature has so far been accomplished, and the generation of electric current by means of a carbon cell seems at present little promising. With metals, as combined in primary cells, the generation of electrical energy is rapid, and this fact has often been neglected by engineers not concerned in electrochemistry. He mentions that a certain manufacturer built a hot-water boiler in which both zinc and copper were exposed to the water, which, of course, resulted in the rapid corrosion of the zinc. Similar mistakes have been made in water meters. Of greatest importance is the subject, however, for shipbuilders. The fact should be taken into account that the electrolytic action changes with the temperature and with the composition of the electrolyte. The speaker then discussed the passive state of metals. He thinks its explanation by means of very thin oxide films on the surface is correct for all practical purposes. For the electrolysis of water for the production of oxygen and hydrogen, iron electrodes may be used, but in such installations it sometimes happens that one single iron anode rapidly corrodes. This is due to a failure of circulation, whereby carbonic oxide accumulates in the positive cell compartment with formation of bicarbonate. As soon as the accumulation of carbonic oxide in the anolyte reaches a certain value the passivity of the iron disappears and the metal corrodes quickly. For this reason good circulation is essential. Heating is also effective, since it makes the carbonic oxide escape as gas before it accumulates to a dangerous degree. The worst case of iron destruction by electrolysis is found in the case of the corrosion of water and gas pipes by the stray currents from the rails of trolley systems when used for the return currents. It seems impossible to avoid this as long as the iron pipes are embedded in wet ground containing carbonic acid. The simplest cure would be to place the pipes so that they are always in dry ground, if that is possible. The author discussed the lead accumulator and the new nickel-iron alkaline accumulator, and then gave a review of electrolytic processes. He emphasized that in the case of metal deposition everything may be accomplished with pure solutions and nothing with impure electrolytes, since the difficulty is not to produce electrically the desired metallic deposit but to provide chemical means by which the electrolyte remains sufficiently pure in commercial operation. He dealt with refining of metals and with the electrolysis of sodium chloride. Asbestos paper and asbestos cloth, stiffened with a coat of cement, are a favorite construction of the diaphragm in this country, while in Germany the diaphragms are generally made of cement, which gets the necessary porosity by the addition of a salt, which is later easily dissolved out of the cement. For sodium chloride electrolysis, sodium chloride is, of course, used as this salt. He then reviewed briefly electric furnaces, electric discharges through gases, silent discharges and endosmosis, and referred to the application of endosmosis to the separation of water from peat. The commercial successes of electrochemical engineering have not been devised at the desk of the theorist, but have been developed by experimental work. Nevertheless, a full knowledge of the theory is most essential for industrial work, since it prevents spending much time and work on useless experiments.

Rapid Electrodeposition of Copper.—This is the subject of a paper of Sherard Cowper-Coles, presented on July 3, before the *Faraday Society*. The paper is here abstracted from advance sheets. The reading of the paper was illustrated by lantern slides and the author exhibited specimens of copper sheets, tubes and wire made by the centrifugal process. The various processes for increasing the current densities in copper deposition by using mechanical means for keeping the copper smooth are classified as follows:

(1) Revolving or moving the cathode, as in the processes of Wilde, Cotsworth, Wylie and Grant. The current density employed is comparatively low.

(2) Burnishing the copper during electrodeposition—the Elmore process. The usual current density is under 20 amps. per square foot.

(3) Insulating the growths on the copper so as to prevent further increase. For example, the Dumoulin process, in which sheepskin and other impregnators are employed. The current density is 35 to 40 amps. per square foot.

(4) Rapid circulation of the electrolyte. Examples: Thofehrn's process, in which impinging jets are caused to play on the surface of a rotating cylinder (C.D. 50 to 100); Graham's process, in which the electrolyte is discharged on to a flat cathode surface (C.D. 300, just under the influence of the jets); Poore's process, in which the solution is sprayed on to the cathode, the stream forming the only electrolytic connection; Dessolle's process, used in Paris for coppering ornamental iron work, and finally, Harrison's, likewise an impingement process. The author considers that impingement processes are not likely to be applied commercially until the amount of solution circulated can be greatly reduced.

(5) Revolving mandrel at a critical speed (centrifugal process). This process has been developed by the author and the latest methods of working are here described. The mandrels are suspended vertically, and are provided with Pelton wheels, which are driven by the electrolyte impinging against them. In the most recent form, however, a tubular vat is used, and hollow mandrels suspended on ball bearings, through the middle of which runs the spindles. These are driven by a worm gearing from below. An 8-foot mandrel has only to be driven at about 50 revolutions per minute.

The author claims the following advantages: The copper is refined and manufactured into sheets or tubes in one operation, and is of a hard nature, similar to that which is cold-rolled; the process is at least ten times faster than any existing electrolytic process; a high current can be employed without deteriorating the quality of the copper; there is no risk of lamination; the plant is simple and free from mechanical complications; the amount of copper locked up for a given output is small compared to other processes; finally, anodes of very impure copper can be used as compared to the anode copper used in other systems. By using a mandrel in which a V-groove has been indented, the spiral deposit that results can easily be pulled away, and then drawn down at once into wire, which can thus be made from crude copper in what is practically one operation. The estimated capital expenditure and cost of working the "centrifugal" process are given in the paper.

A. Stanley Elmore said in the discussion the author had not mentioned heating the electrolyte as a method of accelerating the rate of deposition. He knew of one plant working the Wilde process in which the cathode was rotated rapidly, and the current density was by no means "comparatively low." He gave some further details regarding the Elmore process. The C.D. was occasionally pushed up to 30 amps. per square foot, and has been carried up to over 200 amps. A tensile strength as high as 42 tons per square inch had been attained. By the use of a burnisher the same high C.D. could be used with a less expenditure of power than in other processes, and he claimed that by a combination of burnishing and moderate circulation of electrolyte better and more certain results could be obtained than by the author's "centrifugal" method. He asked how the author proposed to make tubes of less than $2\frac{1}{2}$ inches diameter. Milbourne said that all that was claimed for the "centrifugal" process had been borne out by the results of the trials he had made with a practical plant on the Continent. W. C. Prebble pointed out that in the author's process the circulation of electrolyte took place just at the place where it was most needed, namely, the surface of the cathode. J. V. MacKenzie described an arrangement by which he had used cur-

rent densities up to 45 on electrotyping work. C. J. Steinhart said that capital expenditure was an all-important item; whether that required in the "centrifugal" process remained within the permissible limits remained to be seen. The author had successfully surmounted the "early stage" difficulties of his process. T. C. Cloud criticised the author's theory regarding the growth of nodules. He thought that the toughness of the metal was the result of the particles being drawn out into long fibers as they were being deposited. Cowper-Coles replied to the various points raised. Tubes as small as $2\frac{1}{2}$ inches could not be deposited, but could easily be drawn down.

Nitric Acid from Air.—A very useful review of the various processes which have been devised for the production of nitric acid by electric discharges through air is given by H. R. Carveth and C. L. Rand, in the May issue of the *Sibley Journal of Engineering*. Since the paper is itself a review it cannot be abstracted again. The following table is given of the yields obtained by various experimenters, the figure given being the grams obtained per kw-hour:

1897—Rayleigh	49.1 grams HNO ₃
1900—McDougal and Howles (best yield)	33.8 " "
1902—De Kowalski	55.0 " "
1903—Muthmann and Hofer	70.0 " "
1897—Crookes	74.0 " "
1902—Bradley and Lovejoy	83.0 " "
1904—Birkeland and Eyde	110.0 " "

The calculations of Muthmann and Hofer indicate 157.5 grams of nitric acid per kw-hour as the theoretical maximum efficiency. The above results, which in reality are mere approximations, show that the progress has been satisfactory, and indicates that commercial success may yet be attained. The main improvements which may possibly have an advantageous influence on the yield of nitric acid by the electric flame are: (a) A vessel so designed as to produce the best hot-cold effect. (b) Spark long drawn out. (c) Low current and rapid alternations. (d) Increased pressure. (e) Rapid passage of gases and the introduction of the gases in the correct proportions. (f) Temperature below 1200° C. The authors think it is very probable that indirect methods for the fixation of the nitrogen by means of some intermediate compound such as a nitride or cyanide may be found to be more economical.

EXPERIMENTAL AND THEORETICAL.

Electrode Temperature in Electrolysis.—How important the influence of temperature is in many electrolytic processes is

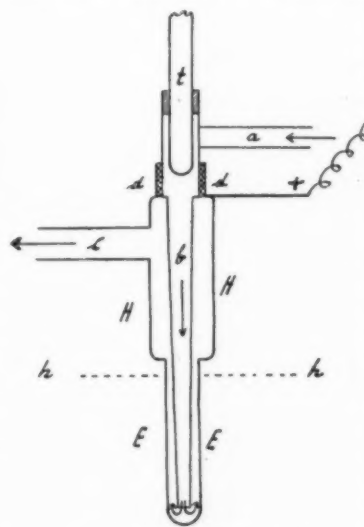


FIG. 1.—COOLED ELECTRODES.

well known, but so far one has been satisfied to regulate the temperature of the electrolyte. W. Moldenhauer, in *Zeit. f. Elektrochemie*, May 26, gives the results of an extensive experimental investigation in which he first investigated the heat set free or consumed at the electrodes during electrolysis. Second, the temperature differences between the electrode and the electrolyte under varying conditions. Finally, he investigated what effect in some elec-

trolytic processes is produced by an artificial cooling of the electrodes. The arrangement is shown in Fig. 1. E is a platinum tube closed at the bottom, which serves as electrode and is suspended into the electrolyte up to the level *hh*. It is soldered to the brass tube H. The cooling water enters through *a*, passes downwards through *b* and then back and leaves through *l*. The thermometer *t* gives the temperature of the cooling water. In the following we give the results obtained with the production of persulphuric acid from sulphuric acid. Berthelot and Elbs and Schoenfeld found that the best conditions are acid of 1.34 to 1.5 specific gravity, high current density and low temperature. The present author first used a concentration of H_2SO_4 of 1.345 specific gravity, with an anodic current density of 100 amp. per square dc. The results are shown in Table I. The efficiency represents the ampere-hour efficiency of the production of $H_2S_2O_8$ in per cent:

TABLE I.

Temperature of electrolyte....	12° C.	12° C.	12° C.
Temperature of anode.....	not cooled	11° C.	0 to 2° C.
Efficiency in per cent.....	54.6	66.4	68.2

He then used more concentrated solutions. His results for an acid of 1.6 specific gravity are given in Table 2:

TABLE II.

Temperature of Anode	Efficiency in Per Cent.
Not Cooled.	55.3
10° C.	36.4
2° C.	9.2

density—and, therefore, the local heating effect—is reduced, considerable quantities of persulphuric acid may be obtained even at higher concentrations.

METALLURGY OF COPPER.

Electric Conductivity of Copper.—A very full and important investigation of the effect of impurities on the electrical conductivity of copper has been made by L. Addicks, the results being given in a paper before the American Institute of Mining Engineers, published in the May issue of the *Bi-monthly Bulletin*. Electrolytic refining has made it possible to produce copper of a very high degree of purity, the metallic impurities averaging but a few thousandths of 1 per cent; oxygen, usually present in the form of compound oxide, bringing the total up to about a tenth of 1 per cent. The author gives in Table I. the amount of various elements which lowers the conductivity several per cent, his researches referring, therefore, to a series of impure coppers rather than alloys.

These numerical results are also given in forms of diagrams by the author, and the author utilizes these curves for plotting a tangent at the zero point of impurity. This gives him the ratio of the lowering of the conductivity to the amount of impurity present, for the case of an infinitely small addition of impurity. This ratio of the percentage lowering in conductivity to the percentage impurity added is given in the column headed "factor" in Table II., where the elements are grouped in their order in the periodic system. In each group the factor decreases with increasing atomic weight.

"The results given in Table I. show what a severe require-

TABLE I.

ELEMENT ADDED.	FIRST SAMPLE.			SECOND SAMPLE.			THIRD SAMPLE.			FOURTH SAMPLE.		
	Added.	Cu.	Cond.	Added.	Cu.	Cond.	Added.	Cu.	Cond.	Added.	Cu.	Cond.
	%	%	%	%	%	%	%	%	%	%	%	%
Aluminium0	99.96	99.6	0.006	99.90	98.6	0.109	99.65	66.8	0.739	99.02	43.5
Antimony.....	0.007	99.96	99.6	0.022	99.94	97.2	0.047	99.86	95.4
Arsenic.....	0.004	99.96	99.6	0.007	99.95	96.8	0.013	99.94	93.2	0.140	99.82	62.3
Bismuth.....	.0	99.96	99.6	0.028	99.93	99.6	0.045	99.91	99.3
Cadmium.....	0	99.96	99.6	0.062	99.90	99.5	0.113	99.87	99.1	0.427	99.55	96.1
Gold.....	0	99.96	99.6	0.089	99.86	98.0	0.149	99.84	98.4	0.317	99.64	96.4
Iron.....	0	99.89	100.5	0.042	99.93	96.8	0.046	99.90	92.9	0.068	99.89	89.6
Lead.....	0	99.96	99.6	0.083	99.82	99.1	0.052	99.86	98.7	0.347	99.56	98.3
Oxygen.....	0.020	99.98	100.7	0.050	99.95	101.4	0.100	99.90	100.5	0.200	99.80	98.0
Phosphorus.....	0	99.96	99.6	0.080	52.3
Silicon.....	0	99.96	99.6	0.007	99.89	99.4	0.007	99.89	99.1	0.042	99.89	99.0
Silver.....	0.003	99.89	100.5	0.137	99.81	100.0	0.340	99.60	98.3	0.503	99.49	97.9
Sulphur.....	0	99.89	100.5	0.053	99.93	100.0	0.135	99.83	99.0	0.236	99.75	98.9
Tellurium.....	0	99.89	100.5	0.065	99.82	100.4	0.181	99.74	100.2	0.405	99.65	68.7
Tin.....	0	99.96	99.6	0.052	99.85	97.6	0.097	99.85	92.7	0.295	99.61	79.8
Zinc.....	0	99.96	99.6	0.048	99.91	98.3	0.095	99.79	96.3

TABLE III.

Current density in amperes per square decimeter..	100.	50.
Efficiency in per cent.....	26.6	63.9

The current density in Table 2 was 100, while the temperature of the electrolyte was 12° C. Table 3 shows how the efficiency depends on the anodic current density.

These results are easily explained. The formation of $H_2S_2O_8$ is due to the combination of two discharged HSO_4 ions. Since sulphuric acid of medium concentration contains at lower temperature more HSO_4 ions than at higher temperature, the rule is that a lower temperature of the electrolyte aids the formation of persulphuric acid. Table 1 shows how this may be easily accomplished by cooling the anode. Further, a more concentrated acid contains many HSO_4 ions already at ordinary temperature. If in this case a strong cooling of the electrode is employed many non-dissociated H_2SO_4 molecules are formed, which must produce a high resistance at the anode on account of the high current density. This causes a high local heat effect which counteracts the formation of persulphuric acid, as shown in Table 2. If, however, the current

TABLE II.

Factor = Percentage Lowering in Conductivity.
Percentage Impurity Added.

Periodic Group.	Element Added.	Factor.	Atomic Weight.	Remarks.
I....	Silver....	6	108	
I....	Gold....	10	197	
II....	Zinc....	30	65	
II....	Cadmium....	9	112	
III....	Aluminium....	500	27	
IV....	Silicon....	70	28	
IV....	Tin.....	67	119	
IV....	Lead....	3	207	Made metal red short.
V....	Phosphorus....	3,000	31	
V....	Arsenic....	720	75	
V....	Antimony....	190	120	
V....	Bismuth....	4	208	Made metal red and cold short.
VI....	Oxygen....	25	16	Large amounts made metal brittle
VI....	Sulphur....	8	32	
VI....	Tellurium....	4	128	Made metal red and cold short.
VII....	Iron.....	140	56	

ment the customary 97 or 98 per cent conductivity specification is, especially as copper is usually associated with arsenic; and when it is considered that the average electrolytic refinery daily passes this requirement by a margin of 2 or 3 per cent, frequently using anodes containing 1 per cent or more of arsenic, it will be appreciated that electrolysis would have come as a refining operation even did copper never carry gold or silver."

RECENT METALLURGICAL PATENTS.

COPPER.

Pyritic Smelting.—With respect to pyritic smelting, two patents of R. Baggaley (789,648, May 9, and 784,651, March 14) require careful interest. His process produces copper mattes without the need of water concentration and calcining and without the use of carbonaceous fuel, or with only a small percentage of such fuel. The inventor claims that in his process "the raw-ore tonnage, without previous concentration, can be quickly smelted, nine-tenths of it discharged as worthless slag, and the remainder converted into matte." Talcky ores and low-grade ores and ores high in iron can be used. The furnace, as shown in Fig. 1, has a lower dissolving and converting portion 2 and twyers 3. The throat 4 is made variable in width by the sliding sections 6, so as to maintain the heat in the proper condition; 7 are the charging doors, 9 the slag tap, or overflow, and 10 the matte tap with its safety-bot; 12 is the safety-botting device for the matte tap-hole; 17 are water-cooled passages.

For the operation of the process the sulphide ores (relatively low in silica and high in matte-making sulphides, such as pyrite, pyrrhotite, and the various sulphides and arsenides of copper) are separated from the more highly silicious ores (whose gangue is granite, quartz). The low-silica sulphide ores are melted in a suitable furnace to produce a low-grade matte that is high in fuel values, i. e., high in sulphur, iron, etc. The matte and slag are discharged into the usual fore-hearth, the slag is separated from the matte, and the clean matte is then drawn off when required for use in the furnace, Fig. 1. This furnace having been heated by a wood fire, or by burning gas or oil, is then charged with a sufficiently large bath of such molten matte to submerge the twyers 3 and to extend above the same to a considerable distance, say up to the level of the slag-spout.

Relatively small charges of the above-described silicious, or sulphide ores, are added to the bath from time to time. through 5 and carbonates, oxides, concentrates, mattes or metal may also be added if available. The blast of air, introduced in great abundance through the twyers, oxidizes the combustible ingredients of the bath (sulphur, iron, etc.) and produces an intense heat. The oxidized iron is fluxed by the silica of the added ore, and forms a floating slag, while the metallic compounds of the ores are dissolved in and become part of the molten matte. As the body of matte is enriched by oxidation its exhausted combustible elements and compounds are replaced by addition of the sulphide ores in solid condition. The

slag is drawn off from time to time through the tap-hole, or overflow 9, so that the surface of the bath is kept clean for the ready dissolving of ore. During the withdrawal of the slag the blast is shut off temporarily by plugging the twyers. For the regulation and control of the volume of blast, delivered into the bath, iron bars are inserted into the twyers.

When a sufficient body of matte of the desired content of values has accumulated in the furnace, a portion of it is tapped off for further treatment through the matte tap-hole 10, and the bath and its heat-producing power are then restored by additions of sulphide ores. It is important that during the progress of the operation the enrichment of the matte should not proceed too rapidly or beyond certain limits. It should never be allowed to exceed 60 per cent in values, and it should preferably not exceed 40 per cent.

R. Baggaley and C. M. Allen (789,133, May 9) refer to the difficulty experienced in pyritic smelting, that when the metallic sulphides melt by a process of liquation out of the silicious shell, the furnace charge is apt to form scaffolds on the walls of the furnace. The slags produced in the converting of copper mattes are necessarily basic, and are specially adapted to dissolve those accretions. The inventors pour the corrosive converter slags, when molten, into the blast furnace, and distribute the same around the walls, thus dissolving all troublesome accretions and at the same time furnishing additional heat. E. W. Lindquist (789,160, May 9) describes details of apparatus for feeding and distributing the molten converter slag into the furnace.

Wet Process.—G. Gin (793,186, June 27), best known by his work in electrometallurgy, has devised a wet process for the extraction of copper from its sulphuretted ores. His scheme is to utilize the sulphurous acid resulting from the roasting of copper ores afterwards for extraction of the copper. The process is based on these three reactions:

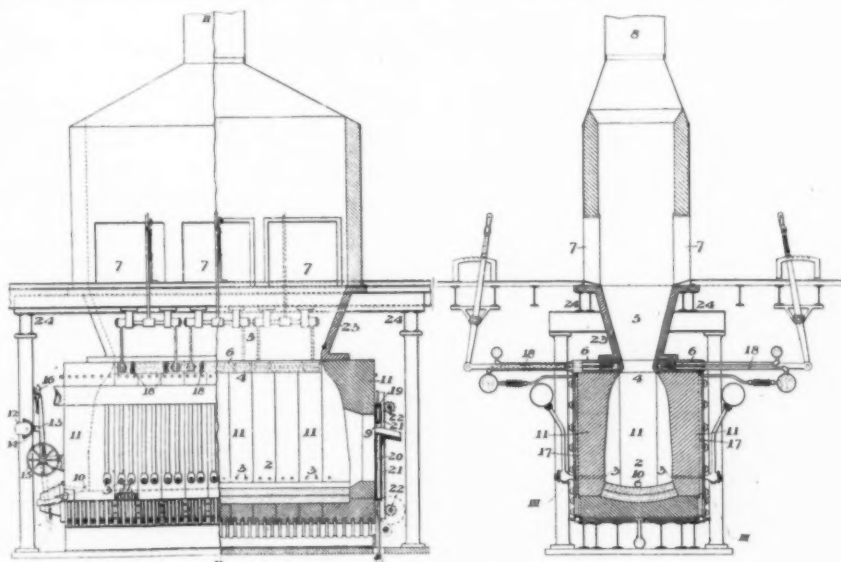
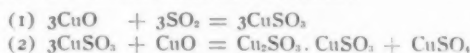
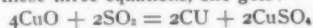


FIG. 1.—FURNACE FOR PYRITIC SMELTING.



By adding these three equations, one gets:



which means that one-half of the copper is obtained in the metallic state and the other half in the form of sulphate. The above reaction (3) requires heating the solution to 180° C. under pressure. In practice the ore is roasted in such a way

that all the sulphide of copper is changed into bioxide or sulphate, and all the iron into peroxide. It is then submitted to methodical lixiviation by means of a solution of sulphurous acid. Reactions (1) and (2) take place, while at the same time iron sulphite and sulphate are formed. The saturated solution of the copper and iron salts is collected by means of a pump in a copper boiler, where it is delivered at a temperature of 180°. At this temperature the iron sulphite and sulphate are quite insoluble and precipitate. Simultaneously reaction (3) takes place. The turbid liquid is then injected under its own pressure into a filter press heated by the vapor circulating in the plates.

ZINC.

Retorts.—In zinc metallurgy, as now carried out, it is necessary to limit the amount of iron, manganese, garnet, fluor-spar, fusible silicates, or the like that may be present in the charge, so that the resultant slag shall contain but a relatively small quantity of these injurious elements. If present in large quantities it is found that the acid walls and bottoms of the retorts are soon destroyed. For this reason the treatment of many kinds of zinciferous ores containing notable percentages of bases, is impracticable. A. L. J. Quenean (789,543, May 9, assigned to New Jersey Zinc Co.) constructs a composite retort. The main body portion is of the usual fire-clay and sand mixture, while the "outer surface" (which may be either the interior or the exterior of the retort) is made up in proper thickness of a mixture of fire-clay and a basic material; here the basic material takes the place of the customary sand, either wholly or in part.

This retort is made in a single operation as follows: The batch of material for the main body portion of the retort is made from a pugged mixture of fire-clay and sand in the usual manner. In preparing the batch for the surface portion of the retort the same fire-clay is used and mixed with an inert or a basic material in granular form (such as chromite, carborundum or the like). This basic material takes the place of all or part of the sand. This mixture of refractory fire-clay and inert or basic material is then pugged in a pug mill to the desired plasticity. It is then stamped in the hammering machine. After the hammering operation it is superimposed upon the preliminarily-stamped clay and sand mixture intended for the main body portion of the retort, and the whole is then thoroughly hammered to a solid wad. This wad is then introduced in the proper position in the chamber of the retort press. The remaining manipulations do not differ from ordinary practice.

IRON AND STEEL.

Liquid Fuel.—A process of smelting iron ore by liquid fuel is described by E. Riveroll (791,577, June 6). The main feature is the deposition of a coating of carbon upon the charge of iron ore and limestone by exposing it to the action of the fuel burning with insufficient oxygen to produce perfect combustion. Fuel thus burned produces a smoky flame, composed of carbon dioxide, plus free carbon, and the latter is deposited in the form of soot. The charge is then subjected to perfect combustion, and the resulting carbon dioxide comes in contact with the carbon deposited on the charge of ore. By this union, at a considerable degree of heat, carbon monoxide gas is formed at the smelting zone, and this gas passes through the charge and robs the ore of its oxygen and leaves the furnace as carbon dioxide. "The ore thus deprived of its oxygen is at the smelting zone reduced, and is from there passed down into the hearth or crucible, where the earthy impurities of the ore unite with the flux, forming a slag, and the iron and slag are separated by gravity, the iron sinking to the bottom and the slag floating on the top of the iron." In a test run of this process, using crude oil only as fuel with a specific gravity of 20.5, the iron ore showed by analysis: Metallic iron, 56.7; silica and alumina, 13.3; sulphur, 1.4. The analysis of the limestone flux was lime, 52.5; silica, 6.3. After smelting the re-

sultant slag consisted of 45 per cent silica and alumina, 50 per cent lime, and 5 per cent iron and other elements.

Open-Hearth Operation.—H. Knoth has formerly (712,389, Oct. 28, 1902) patented a process of manufacturing steel, wherein the furnace contents are tapped into a ladle in which a portion of the steel is reserved and mixed with the molten pig metal with which the ladle is filled, after its slag and the greater part of its steel contents have been poured off. The ladle, with its mixed charge, is then run back and charged into the same furnace or any other that may be ready to receive it. The object was twofold: first, to reduce the time required to prepare the heats in the furnace for lapping and to avoid the cost of scrap charged with the pig iron. This process still involved waste and delay in the production and maintenance of a highly basic slag capable of taking up the metalloids in the bath. The same inventor now proposes the following process (788,650, May 2):

The unpurified pig metal, constituting the initial charge, when high in silicon, is subjected in a liquid state to an oxygen-yielding blast in an acid-lined vessel and the silicon and part of the carbon content of the metal reduced. (If the pig is sufficiently low in silicon this converter step is omitted.) The desiliconized metal is then introduced into an open-hearth furnace and there treated according to the usual methods, and in the presence of a highly-basic slag to remove the phosphorus and reduce the carbon to the desired point. The slag which results from this treatment is drawn off with the molten steel into the same or a separate ladle, from which it is reintroduced back into the same or any other furnace, as long as it retains its basic qualities. The lower the metals charged are in silicon and phosphorus, the greater is the saving.

Both processes combined are worked as follows: An initial heat of metal and slag is prepared and treated in an open-hearth furnace. The furnace contents are tapped into a ladle. Two-thirds of the purified metal are poured into molds; one-third is recharged into the furnace with new unpurified material (which may have been previously treated in a Bessemer converter). The reserve of slag is then fed back into the furnace.

Purifying Pig.—J. B. Nan (786,048, March 28) pours liquid pig metal upon a mass of broken pieces of solid oxidizing material (iron ore) in such manner that the liquid metal is broken up and descends in small streams through the spaces between and in contact with the pieces of ore. A bath of liquid metal thus accumulates in which the pieces of oxidizing material are maintained immersed until the reaction has proceeded to the desired point. Several arrangements of apparatus are described for this purpose.

Charcoal Iron.—J. J. Hudson (785,002, March 14) makes charcoal iron in a furnace heated by means of oil, gas or hydrocarbon vapor. In this furnace the metal is melted in the presence of charcoal, while at the same time blasts of air are introduced into the metal for the purpose of boiling it. The metal is thereby refined by means of the charcoal, air blast, and other source of heat, eliminating sulphur, phosphorus, silicon, etc. The metal filters through the charcoal, and the latter consumes the impurities as they are eliminated from the molten mass. The mass is decarburized until the analysis of the molten metal shows the quality of iron required. The metal is then drawn directly into a suitable mold, from which it may be taken as soon as it has received a surface chill, given a wash heat in a suitable furnace or oven, and then passed directly to the mills to be rolled into skelp, plates, or other shapes. The inventor proposes to use his process for the refining of mild steel by charcoal, so as to render it available for all purposes for which charcoal iron of the best grade is available.

Iron Sand, Fire Ore, Flue Dust, etc.—M. Moore and T. J. Haskett (791,928, June 6, and 792,440, June 13) treat "magnetic iron sands," or finely crushed ferruginous ores, as follows: They first concentrate the ore and separate from it any silica

or earthy materials. The ore is then treated in a tower containing various chambers. While passing through the first chamber the ore is subjected to the action of heat, produced by the admittance of air with waste carbonic, oxide or hydrocarbon gas issuing from a second chamber. The ore is then passed through this second chamber, and is here subjected to the progressive reducing action of those gases alone. The result is that all the oxygen is removed from the ore, which is thereby reduced to the metallic state, but is still in a finely-divided condition. Without coming into contact with an oxidizing atmosphere, the metal is then passed into a Siemens, or other gas furnace, and immersed in a bath of molten metal or slag, which is always maintained therein, and in this furnace it is fused and "balled up" as wrought iron or converted into molten steel. During the time the charge is being delivered to the gas furnace the supply of atmospheric air to the gas furnace is cut off.

D. Baker and W. W. Hearne (788,813, May 21) prepare magnetic concentrates of fine iron ores, such as pyrites residuum, pyrites cinders, etc., as follows for the treatment in the blast furnace: If the material has a large sulphur content it is first desulphurized. For this purpose it is fed into a long rotary kiln, where it is agitated at a temperature low enough to prevent the sulphide of iron from smelting and at the same time maintaining strongly oxidizing conditions. When the desulphurization is complete the material reaches the zone of highest temperature in the kiln where it is agglomerated. No fluxing material is used, but those foreign particles of alumina, silica, lime, magnesia, etc., which are intimately combined with the iron oxide in the ore are utilized. These are fused together with the iron oxide, resulting in an agglomeration of the ore.

C. S. Price (794,152 and 794,153, July 4) prepares flue-dust for the treatment in the blast furnace by mixing 4 to 8 parts of flue dust with 1 part of ground clay in a pug mill. This mixture is moistened with sufficient water to agglomerate it into lumps, which are charged into the blast furnace. The clay is said to be useful in serving to increase the quantity or regulate the quality of cinder, as required in many cases.

Cleaning Furnace Gases.—For the utilization of blast furnace gases it is of utmost importance to thoroughly clean them. Three recent patents refer to this subject. J. S. Oursler (791,160, May 30) employs a gas-washer adapted to cause the gas to pass in a sheet against and into a stream of water, sprays of water being employed to aid in the washing and cooling of the gas. The foreign matter is collected in dust pockets.

W. Schwarz (793,544, June 27) employs an apparatus belonging to that class of purifiers in which water and centrifugal force are used. The special feature is that the purifying effect is confined to the periphery, *i. e.*, to annular spaces contiguous to the internal surface of a stationary casing of cylindrical form.

J. Shields (793,745, July 4) conducts the dusty gases beneath or into the midst of a constantly-replenished heat of porous or granular material. The gases pass upward through the heap, and the filtering material collects the dust, and is gradually drawn away from the bottom of the heap, while the top is constantly replenished with fresh material. The gases thus encounter cleaner and cleaner material as they rise. Such a filter is stated to be specially useful in the treatment of the gases issuing from pyrites-burners. Suitable filtering materials are coke, sand, crushed quartz, granulated slag, pumice, etc. The arrangement is shown in Fig. 2. The gases are conducted into the heap by pipe *c* through downwardly-projecting orifices *c'*, situated beneath a horizontal disc *d*, which creates a cavity in the middle of the heap. The gases pass off through *l*.

METALLURGICAL FURNACES.

Roasting Furnace.—For the treatment of ores which require a comparatively long exposure to heat, it is economical to materially increase the diameter of the ordinary McDougall

furnace to secure an increased area for the hearth. With the usual construction this involves a difficulty in so far as the ore tends to accumulate toward the center of the hearth and to choke the furnace. In order to dispose of it as fast as it accumulates, F. Klepetko (793,939, July 4) provides each rabble-arm with a series of rakes whose depths increase as they approach the center of the furnace, so that the radically-effective sweeping surface increases toward the center. The same inventor (792,053, June 13) modifies a detail of construction of

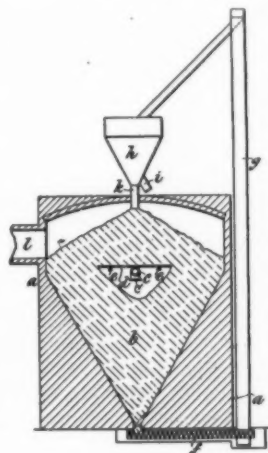


FIG. 2.—APPARATUS FOR CLEANING GASES.

the rabble arms described in one of his former patents. For cooling purposes these arms are made hollow, and the inventor now provides them with ribs on the inner surface, following the general longitudinal direction of the arm. C. S. Repath and F. E. Marcy (794,118, July 4) use a central hollow shaft from which radiate the hollow rabble arms; air for cooling the shaft and arms is taken in at the top of the central shaft, either by natural or forced draft, from where it passes into the arms, and from there it may be discharged into the charge, so that the air answers the twofold purpose of cooling and furnishing oxygen to the charge. The connection between central shaft and rabble arms is as follows: Passed transversely through the walls of the shaft in earth-hearth is a T-bar, eye-beam, or the like, procured to the shaft walls by means of angle-plates. The projecting portions of the T-bars serve as support for the interchangeable hollow rabble arms, which are simply slipped over the support and their ends keyed by a wedge.

A. M. Beam (793,816, July 4) patents mechanical details of construction of a roasting furnace in which the ore travels through a rotating cylinder, which is subjected to the products of combustion from two fire-boxes. There are three concentric cylinders; the outer and middle one are detachably secured together, so as to rotate in unison, while the central cylinder is connected to the middle one by a series of integral partitions. The space between the middle and the central cylinders forms the ore chamber, while the annular space on the outside and the cylindrical space inside are flues, through which the products of combustion pass from the combustion chambers to the stacks.

Gas Furnace.—J. C. Swindell (787,131, April 11) patents details of a compact arrangement of the combustion chamber of a gas furnace, surrounded on its top and sides by channels in such a way that the incoming air is thoroughly heated by the heat imparted to the walls of the combustion chamber and by the heat of the waste gases. G. L. Fogler (786,746, April 4) heats the gas supplied to the furnace before it is fed into it; for this purpose he uses the waste gases of his furnace, which are, of course, first collected and purified. A. Kurzwernhart (786,770, April 4) utilizes the waste gases in a Siemens regenerative furnace to force the combustible gas into the furnace.

Hydrocarbon Furnace.—C. E. Glafke (790,825, May 23) uses a revolving, inclined, longitudinal smelting chamber into which the ore is fed at the higher end; at the same end the burner is provided, fed with hydrocarbon fuel. The other (lower) end of the chamber is partially closed by a baffle and projects into the stack, in the bottom of which is the molten ore receptacle.

G. L. Bourne (793,174, June 27) endeavors to increase the efficiency of furnaces using crude petroleum or fuel oil as a

fuel. For this purpose he introduces oil and air into a primary combustion chamber, and there volatilizes and converts the oil into a complete gaseous state, and then introduces a blast of fresh air to supply the necessary additional oxygen to provide a perfect combustion in a secondary combustion chamber. The heating chamber is located adjacent to the secondary combustion chamber.

Various Furnace Constructions.—J. L. Lansing (792,223, June 13) patents a method of roasting telluride ores. In order to prevent any considerable portion of the values from being lost in the form of tellurium fumes, he roasts the ore in closed furnaces which are connected with a condensing chamber. The latter contains cold water through which the telluric fumes and gases must pass before their discharge into the open air.

W. S. Rockwell (792,169, June 13) patents a combination of two rotary converting chambers, independently rotatable and located end to end in close contact with each other. While the metal in one chamber is being molten, ingots are placed in the other chamber and are subjected there to a preliminary heating, the products of combustion being passed over from the first chamber to the second. After the metal is tapped from the first chamber the metal in the second chamber is melted and new ingots are heated in the first chamber.

In order to avoid tilting, J. Morat (784,956, March 14) mounts the furnace centrally so as to be revoluble on a support, the axial line of the vessel being inclined to the vertical. The vessel is revolved on this angular axis so that certain charging or discharging openings in the side of the vessel are brought either below the level of the molten metal or above it.

W. E. Williams (792,642, June 20) uses an almost spherical melting furnace with a single opening, mounted so as to be revolved in two planes. There is provided a set of fixed concentric flues for supplying air and fuel and discharging the furnace gases, and means for rotating the furnace while so connected with the flues, although otherwise closed. The intention is to expose continuously the contents of the furnace to the freshly-heated portion of the lining that comes underneath it as the furnace revolves, while the melted metal keeps flowing down to the bottom and the unmelted parts are carried up to the sides to be exposed above the melted portion.

C. C. Medbery (788,546, May 2) patents a furnace with a cylindrical revoluble heating chamber supported and controlled by anti-friction rollers. Means are provided for tilting to facilitate charging and pouring, and for adapting the heating chamber to be used as a converter and for reducing and refining operations, also air blast and fuel-supplying pipes and devices, etc.

Miscellaneous Details of Furnace Construction.—P. Healey (786,121, March 21) endeavors to improve the construction of smelting furnaces in which the lower portion of the stack is provided with double walls, forming an annular jacket for water or air by which that portion of the stack around the zone of fusion is kept cool. The feature of the present invention is that neither water nor air alone is used, but a mixture of the two in the form of an atomized spray.

J. W. Seaver (792,735, June 20, assigned to Wellman-Seaver-Morgan Company) patents a furnace-filling apparatus for feeding rectangular elliptical or other elongated furnaces and smelters. He provides two traveling covers connected together and spaced apart and an endless conveyor mounted upon one cover to feed the charge between the two covers; while a traveling lorry runs on the cover carrying the conveyor and discharges on to the latter.

J. B. Ladd and D. Baker (785,311, March 21) patent a bell to secure an even distribution of stock in a blast furnace. The bell comprises essentially a number of concentric sections movable relatively to each other.

W. Kennedy (792,047, June 13) patents apparatus for changing the bell and bell-ring of blast furnaces. The usual bell-operating mechanism is employed for raising the bell and the

hopper, which has its lower portion formed by a bell-ring, to such a height that the removal of the bell-ring and bell from position over the furnace is possible and new parts may be substituted.

BOOK REVIEWS.

ÜBER DIE FERROMAGNETISCHEN EIGENSCHAFTEN VON LEGIERUNGEN UNMAGNETISCHER METALLE. By Fr. Heusler; 64 pages, 13 figures, 8 tables and 3 curve sheets. Marburg; N. G. Elwert.

This little volume on the magnetic properties of alloys of non-magnetic metals gives a brief outline of the principles of magnetism with special reference to the recent important researches of the author upon magnetic alloys.

The principal experimental results with alloys of manganese, aluminium and copper are summed up in two short chapters. It appears that the best magnetic effects are obtainable with about equal molecular weights of manganese and aluminium. This may be called the Mn—Al alloy. The greater the percentage of this alloy in copper, the higher the final resulting magnetic susceptibility. The highest flux density recorded is 6480 gauss.

The pamphlet will have special interest for students of physics and magnetism. It forms a separable part, or section, of a larger treatise, entitled "Schriften der Gesellschaft zur Beförderung der gesammten Naturwissenschaften zu Marburg." "Publications of the Marburg Society for the Advancement of Natural Philosophy."

Diagonal Concentrating Table.

In concentrating mineral particles on a table three forces are employed, viz.: gravity, reciprocating motion and flow of wash water.

Gravity acts in a vertical direction on the mineral particles, forcing them downward, the heavier particles occupying the lower strata.

The reciprocating motion of the table is a horizontal force, and tends to bring the particles of ore forward in the direction of the line of reciprocation. The flow of wash water is also a horizontal force, but transverse to the reciprocating motion. Thus, the reciprocating motion and the flow of wash water act in nearly transverse horizontal lines, so that the resultant of these two forces is a line diagonal to both. Consequently, the material on the table tends to travel in a diagonal direction. In consequence of this general tendency the Overstrom concentrating table, built by the Allis-Chalmers Co., Milwaukee, Wis., is built diagonally to the reciprocating

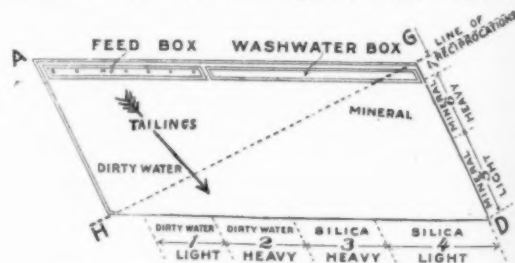


FIG. 1.—DIAGRAM OF ACTION OF DIAGONAL TABLE.

force and to the flow of wash water as indicated in Fig. 1.

As a matter of fact the whole material would travel in a diagonal direction only if it was uniform in composition, since, in this case, each particle would be acted upon with equal force by the two horizontally acting forces. In reality, however, we have a mixture of mineral particles of higher specific gravity with gangue or silica of lower specific gravity.

The reciprocating motion of the table has a greater action on the particles of mineral, which are closer in proximity to the table surface than on the lighter gangue or silica which remains on top of the mineral. On the other hand, the flow of wash water produces a greater movement of the gangue or silica than of the mineral particles, and washes the particles of the lighter gangue a greater distance in a given time. As the gangue is on the top of the mineral it is acted upon more freely by the wash water until a final separation takes place. Thus, it is that while uniform material of medium specific

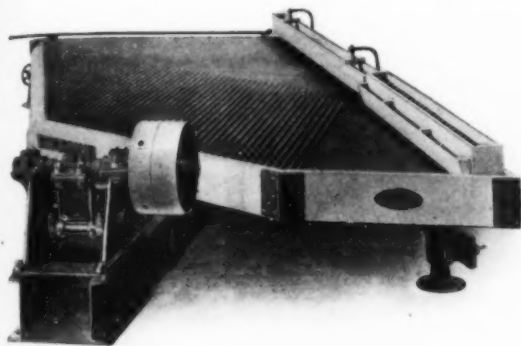


FIG. 2.—OVERSTROM CONCENTRATING TABLE.

gravity would tend to travel in a straight line towards D in Fig. 1, the non-uniform material is separated, the heavier mineral being moved by the reciprocating action away from the straight line towards GD, while the lighter gangue is deviated by the wash water to the other side towards HD.

The diagonal construction of the table has some very important consequences. First, nearly the whole available surface of the diagonal table is effective surface. The large effective surface diagonal to the reciprocating motion and to the flow of wash water results in a largely increased capacity and cleaner separation, owing to the greater transverse distance the material travels on the table.

Further, this construction enables the use of a long feed-box. Attached to the table, it receives the reciprocating motion of the table, distributes the feed evenly over a large surface, and also, as a result of the shaking, the box becomes an especially efficient sizer in itself. The Overstrom long feed box is placed on the side of the table along which the mineral travels, i. e., along the natural line the mineral takes when acted upon by the reciprocating motion and the wash water. The feed-box is also constructed so as to give a partial concentration. On a mixed feed it will collect the large, round mineral, which has a tendency to roll if left on the table direct, and discharge it through an opening in the forward end of the box, leaving this material high up out of danger. There is also a saving of clean wash water as a result of using a long feed-box.

Fig. 1 shows diagrammatically the separation of the material into different zones. At the point D, which is the point of final separation, both the mineral particles and the silica come off light, with practically no mixture of mineral and silica, but with a clean separation, and consequently but little middlings. (This is an important difference from the action of

rectangular concentrating tables, where at the point D both the mineral particles and the silica come off heavy and thick with a consequent mixture and an absolute necessity for the rehandling of this mixed material, which is usually called middlings.) The arrow in Fig. 1 shows the direction of the movement of the dirty water across the table. Otherwise this diagram is self explanatory.

Fig. 2 gives an external view of the Overstrom concentrator from the head motion end. In its latest improved design the concentrator embodies several important features. The table is made of structural steel and iron throughout, with the exception of the table top, which is made of California redwood. This construction gives great rigidity of frame with a greatly increased smoothness of table action. The table has been provided with a newly designed head motion, so as to advance the mineral particles more rapidly. To impart the reciprocating motion, rocking arms are employed and a most perfect table action is produced. The table top rests on four long rollers, which extend the whole width of the table and give more than 24 feet of bearing surface. No lubrication is needed, and the action is so that the rollers cannot wear flat. Concerning the details of the construction, as well as erection and operation, the reader may be referred to a fully illustrated pamphlet just issued by the Allis-Chalmers Co. on the Overstrom table (catalogue No. 124).

Fig. 3 shows an installation of Overstrom concentrators in the mill of the Baltic Mining Co., Redridge, Mich.

Exhibits at Boston Electrical Exhibition.

In the Boston Electrical Exhibition, held from July 15 to 22, the General Electric Company had one of the largest spaces in the center of the main floor, covering about 1300 square feet. The principal feature of this exhibit, and probably the newest thing in the entire exhibition, was the new G. E. $2\frac{1}{2}$ -watt incandescent lamp. The company's exhibit was brilliantly illuminated with these units. The lamp should be of special interest

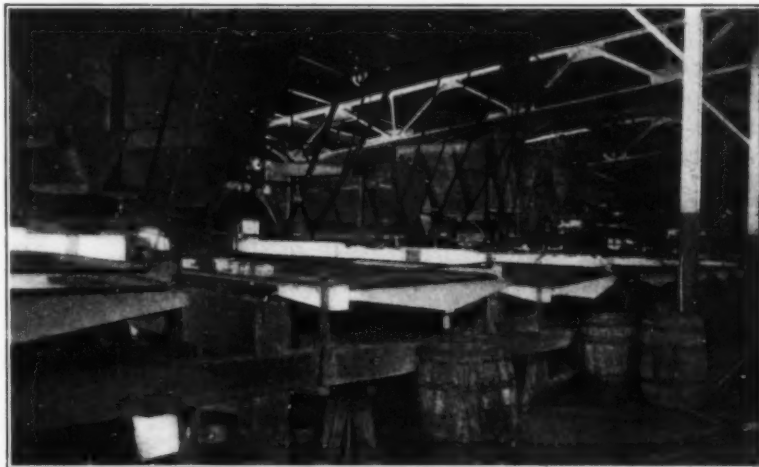


FIG. 3.—OVERSTROM TABLES IN MILL OF BALTIC MINING CO.

to our readers in view of the rôle which the electric furnace plays in the manufacture of the filament. (See the paper of J. W. Howell, p. 252 of our last issue.)

This is the first exhibition to the general public where this carbon filament has been shown, and the Electrical Contractors, under whose auspices the exhibition was held, may well feel proud to have their show characterized by such an important event. The new metallized carbon filament lamp is made in three sizes, Nos. 3, 4 and 5, the watt consumption being 125,

187½ and 250 watts per lamp, respectively. These lamps are used with a special line of Holophane pagoda reflectors, which are made in two forms, one which distributes the light below the horizontal plane, and the other which concentrates the candle power in a downward direction. The following table shows the ratings of these lamps:

Type Number.	Total Watt per Lamp.	D istributed Candle-Power in Downward Direction with "D" Form of Reflector.	Concentrated Candle-Power in Downward Direction with "C" Form of Reflector.
3	125	70	120
4	187½	105	180
5	250	140	240

They certainly present an attractive appearance, and as the actual gain is 20 per cent greater efficiency over the best obtained from the highest efficiency incandescent lamps of the past on an equal basis of life, they are sure to meet with a hearty reception. They are particularly suitable for use where



MERCURY ARC TOWER.

the arc lamp is too large and the ordinary incandescent too small.

Another exhibit of the company of special interest to our readers was a mercury arc rectifier operated on 220-volt, single-phase, which was converted to 110 volts, direct current, the load being a circuit of the new incandescent lamps. This rectifier has a capacity of 30 amps. and was in constant operation. The application to storage battery charging and for electrolytic purposes in general is obvious.

A collection of small motors of various types was shown. This included alternating and direct-current motors. A new G. E. electric flat-iron also made its appearance here for the first time, and naturally attracted a great deal of attention from the public.

In attendance at the exhibit were Mr. C. B. Davis, Boston manager; Mr. J. P. Felton, supply manager, Boston office; Mr. O. F. Brastow, L. E. Smith and J. A. Wilson, Boston; Mr. E. M. Kinney, Schenectady, N. Y.; Mr. W. M. Wright, New

York, and Mr. F. H. Gale, advertising manager. Other callers during the week were M. T. Beran, New York, supply manager, and Mr. D. R. Bullen, Schenectady.

Mercury Arc at the Elks Carnival at Buffalo.

The mercury arc lamp is of special interest to our readers on account of its well-known rectifying effect. It will undoubtedly be used to a large extent in future for electrolytic purposes and in connection with storage batteries, when only an alternating current supply is available. The following note refers to the use of the mercury arc lamp for producing special and very beautiful effects in street lighting.

The entire business part of Buffalo was in gala attire in honor of the nineteenth reunion of Elks in the week of July 10. But if the daylight scene was gorgeous, it was far excelled by the electrical decorations which literally turned the night into day.

Through the energy and resourcefulness of General Manager Charles R. Huntley, of the Buffalo General Electric Co., the city streets presented a night scene of splendor probably never before equaled in similar celebrations. Niagara electric power had been freely drawn upon, and was everywhere in evidence, through incandescent lamps festooned across the fronts of buildings in "welcome" signs, and in elaborate special designs embodying the characteristic Elk emblem.

The unique feature of this veritable festival of light was a "Mercury Arc Tower," located in Shelton Square. This was a fluted column of "Staff," 75 feet high, studded with 100 mercury arc lamps. Standing alone in its splendor, this tower produced a most startling effect, due largely to the characteristic greenish light of the mercury arc lamps. It stood out conspicuously in the mass of many colored incandescents covering the fronts of the buildings on the Square and the radiating streets.

The mercury arc lamps were operated in two series of 50 lights each from 4-amp. Brush arc machines, located in the Wilkinson Street station. As each lamp consumes only 166 watts, the operating expense was extremely low for so brilliant a spectacle. They are of the standard type for outdoor lighting, as developed under the direction of Dr. C. P. Steinmetz, and manufactured by the General Electric Company.

The new use of the mercury arc lamp on a large scale marks a radical step in electric lighting for decorative purposes. The incandescent lamp reached its climax for general illumination and for decorative effects at the Pan American Exposition four years ago in Buffalo, and it is interesting to note that the latest development in this branch of the art should be successfully tried in the same city and using also Niagara power.

This luminous tower was a special contribution to the carnival by the Niagara Power Co., the Cataract Power & Conduit Co., and the Buffalo General Electric Co. Mr. Huntley conceived the idea, and designs were prepared by the illuminating engineers of the General Electric Co., and this company furnished the material and superintended the installation.

Industrial Notes.

Slag Ladles and Cars.—A recent illustrated pamphlet of the Wellman-Seaver-Morgan Co., of Cleveland, deals with Dewhurst patent slag ladles and cars, which have already been extensively adopted in leading plants in Great Britain. They are either built for side tipping or end tipping as required. The ladles are tipped by a pull of the locomotive on the tipping chain, the locomotive pulling away from the ladle while dumping. The ladles are special shape, to facilitate the stripping of the slag and skull. Hence, in dumping, the skull follows the molten slag, requiring no hand work for its removal. The whole of the work is handled by the locomotive

and its crew. The chief point is the great simplicity of design, since there is no complicated tipping mechanism, no steam or air cylinders, nor gearing, no tipping poles, no hand work in removing skull. The reduction in the amount of hand labor not only reduces the cost of operation, but also facilitates the work. The cars consequently make quicker trips and fewer cars are required to take care of the work.

Small Direct-Current Motors.—The nicely illustrated folder No. 4039, of the Westinghouse Electric & Manufacturing Co. deals with direct-current motors, type R. This type covers small motors, in sizes from 1-6 to 1½ hp., and suitable for almost any direct-current service within their capacity. They are specially useful for driving small machine tools, small pumps, mixers, stirrers, labeling machines, etc.

We have received from the *Waterbury Brass Co.*, of Waterbury, Conn., their catalogue on brass, copper, German silver in sheets, rolls, plates, circles, wire, rods, brazed tubing and iron lined tubing. The catalogue gives first concise definitions of various kinds of brass and some other alloys, and notes on tempers, anneals and gauges. Then follow notes and tables on various brasses in sheet, roll and wire form; tables on the electrical properties of pure copper wire, "white metal" wire, and German silver wire; comparative tables of standard sheet and wire gauges (B. & S., Birmingham, inches, mm.); tables of weight and gauges of brass wire, weights per linear foot of brass and copper rods, weights per square foot of copper and brass sheets, weights of Muntz metal and Tobin bronze plates and rods, etc.; tables on B. & S. drill wire gauge; very full tables on weight per foot of seamless brass tubes and seamless copper tubes; tables of iron pipe sizes of seamless drawn brass and copper tubing; brazed brass tubing, and various useful tables.

The New Denver Mint.—Messrs. GEO. D. FEIDT & Co., of Philadelphia, have received an order from the new Denver Mint for a general outfit of chemicals and chemical apparatus, including certain electric furnaces and heating apparatus, for use in the melting and refining department of the mint.

The De La Vergne Machine Company, of New York City, have issued a small illustrated folder on Koerting gas engines. It contains an interesting picture of a plant with 2000-hp gas engines driving blast-furnace blowing cylinders. These engines operate with blast-furnace gas of from 80 to 100 B. T. U. per cubic foot.

CYANIDE.—In view of the steadily increasing demand for cyanide, especially in gold metallurgy, the United Alkali Co., Ltd., of Liverpool, has perfected processes for the manufacture of high-strength cyanide of sodium, for which as special advantages are claimed high cyanogen strength, great solubility, absolute freedom from sulphides, high extractive value and economy in freight charges. The cyanogen content is equivalent to from 105 to 110 per cent, and a higher grade from 120 to 130 per cent, compared with the ordinary potassium cyanide. It contains a proportion of carbonate of soda, which favors the action of the cyanogen in the solution by assisting to neutralize any acidity. This sodium cyanide is already in successful use in the gold fields of Australia, South Africa and India, and is now introduced in this country by Messrs. James Lee & Co. (founded 1825), 76 William Street, New York.

CURTIS STEAM TURBINE TESTS.—In view of the fundamental importance of the power problem in metallurgical and electrochemical operations, the following results of tests of a 2000-kw Curtis steam turbine, made by the GENERAL ELECTRIC Co., are very important. The turbine is a four-stage machine, designed in 1903, and recently changed in a few particulars as a result of experiments conducted during the past year. The machine as tested conforms as nearly as possible to the standard four-stage machines now produced by the General Electric Co., but is less efficient, since the changes made have been confined to the buckets, while other desirable changes could not

be made without rebuilding the machine entirely. The machine operates at 900 r. p. m. The tests were made by Messrs. F. Sargent and L. A. Ferguson, of Chicago, by the most accurate methods available.

	Full Load.	Half Load.	Quarter Load.	Zero Load.
Duration of test in hours.....	1.25	0.91	1.	1.33
Steam pressure (gauge) in lb.....	166.3	170.2	155.5	154.3
Back pressure (absolute) in mercury.....	1.49	1.40	1.45	1.85
Superheat deg. F.....	207.	120.	204.	156
Load in kilowatts.....	2037.7	1066.7	535.	
Steam consumption per K.W. hour in lbs.....	15.02	16.31	18.09	15.105

LOW-SPEED GENERATORS FOR ELECTROLYTIC REFINERY.—The output of our electrolytic refineries is constantly increasing. This flourishing condition is indicated by an order just placed with the CROCKER-WHEELER Co., of Ampere, N. J., through its New York branch. The order calls for two Nr. 898 Crocker-Wheeler engine-type, direct-current generators, with a capacity of 10,000 amps. and 105 volts. These will be special machines with twenty-two poles. The speed will be 100 r. p. m. They will be installed at Carteret, N. J., in a plant where there are at present four Crocker-Wheeler generators of varying capacity up to 750 kw.

THE THERMIT PROCESS IN AMERICAN PRACTICE.—A suggestive and interesting paper on this subject was presented to the American Society for Testing Materials by Mr. Ernest Stütz, general manager of the Goldschmidt Thermit Company. The paper has recently been issued in pamphlet form, and is especially interesting since it shows how the thermit process of Dr. Hans Goldschmidt has been introduced for many purposes in this country. The first thermit was made in this country a year ago, and the success of its applications for many different purposes is attributed by Mr. Stütz very modestly, but at the same time undoubtedly with much justification, to the fostering care and ingenuity of American engineers. The essential point of the Goldschmidt thermit process is that it enables one to get highly superheated steel of any desired quality anywhere in anything like half a minute. The essential characteristic of thermit is that it welds by fusion, and by reason of this fact calls for the foundrymen's experience more than the blacksmith's. Its success depends on the proper material, shape and condition of the mould. Concerning this some useful hints are given by Mr. Stütz. The important problem of the welding of broken locomotive rails has been investigated by some thirty railroads in this country with more or less success. In all there are records of thirty engines with welded frames that have been in service for three months or longer. Another operation of interest to railroad men is the welding of spokes of drivers. In marine engineering, notable success has been obtained in making repairs with thermit, among others, by Mr. Des Anges, superintendent of the floating department of the Long Island Railroad. Some important repairs of great iron castings are also reported by Mr. Stütz. The process of rail welding for electric traction systems has been investigated in about thirty different American cities in actual operation, and about 5000 joints have been put in up to date. That steel foundries should have been first to recognize the possibilities of liquid steel that can be produced anywhere in half a minute goes without saying. There are already several of the largest with whom thermit is now as much a necessity as foundry sand.

MEASURING AND RECORDING INSTRUMENTS FOR CHEMICAL AND ELECTRICAL WORK.—We have received the catalogue of accurate measuring and recording instruments for all scientific and industrial purposes, made by M. JULES RICHARD, of Paris, represented in this country by Mr. Ernest H. Du Vivier, 14 Church Street, New York City. The catalogue deals especially with recording gauges, recording thermometers, pyrometers, ammeters and voltmeters. The apparatus are made for general work as well as for special purposes in different industries. Some of the instruments made for use in chemical works, sugar factories and refineries, etc., etc., are quite interesting. For instance, recording thermometers are made in a great many types for various purposes. Sometimes it is not

only necessary to obtain a chart of the temperature, but an attachment is advisable by which, with the aid of an electric bell, the workingman is warned that a dangerous temperature is being reached. This is the case, for instance, in the distillation of ammoniacal products, for which purpose a recording instrument is made having a dial 9 inches in diameter and a tube covered with lead $6\frac{1}{2}$ feet long. It is graduated between 30° and 130° C., or 86° and 266° F., and is fitted with a device to give warning when the danger points are reached.

Orders have been received by the WESTINGHOUSE ELECTRIC & MFG. Co., from Nelson Morris & Co., for nineteen type CCL induction motors varying in sizes from 5 to 50 hp, and totaling 410 hp, and for eight motors of the same type from the Decatur Car Wheel & Mfg. Co., Birmingham, Ala. These are the latest design of induction motors which have been built by this company, and have just recently been placed on the market. On the same day an order was entered from the Illinois Steel Co. for eighty direct-current motors having an aggregate capacity of 3920 hp. These motors are all of the series-wound type, and range from 30 to 75 hp. The Rumford Falls Power Co., of Rumford Falls, Me., will install one 550-kw and one 800-kw alternating-current generators, which will be direct connected to waterwheels. The order includes two 540-hp variable-speed induction motors, and 120 type OD transformers for use on the distributing lines.

Personal.

Mr. B. H. WARREN, president of the Allis-Chalmers Co., has returned by the Baltic from England, after a prolonged stay of three or four months abroad.

Dr. J. W. RICHARDS, of Lehigh University, sailed on July 22 for Europe, to spend six weeks in Great Britain. He will be back in time for the Bethlehem meeting of the American Electrochemical Society.

We had a pleasant visit from Mr. ADOLPH CHALAS, eldest son of the senior of Chalas & Sons, in London. Mr. Chalas is visiting this country to study electrochemical and metallurgical developments.

Prof. HENRY S. CARHART sailed on the Graf. Waldersee to London, upon a special invitation, to join the official party of the British Association for the Advancement of Science, on their trip to South Africa, where the meeting of the association is to be held in Cape Town and Johannesburg. On his return he will attend a conference, called by the German Reichsanstalt, preliminary to the meeting of the International Committee on Units, which will meet in a year or two, according to the plan adopted at the St. Louis Congress. The preliminary meeting will be attended by the official heads of the United States Bureau of Standards, the (British) National Physical Laboratory and the German Reichsanstalt, and by the following four gentlemen: Prof. Carhart, Prof. Kohlrausch, Prof. Mascart and Lord Rayleigh.

Mr. W. M. PROBASCO has resigned as assistant to the president of the McGraw Publishing Co., to become the vice-president of the Search Light Publishing Co., of New York City. This company owns a most elaborate pile of clippings, articles and pictures on all subjects, classified and kept up to date for the use of all who require prompt information on any subject. As part of its plan the company publishes a weekly called the *Search Light*. Other departments of the company are the publication of books and the handling of the publishing and advertising accounts of important railway, manufacturing and engineering firms. Mr. Probasco was formerly assistant general manager of the Westinghouse Co.'s publishing, advertising and exhibition interests, and in this capacity he designed

and organized the Westinghouse exhibits at the St. Louis World's Fair, which received the grand prize.

We received a pleasant visit recently from Dr. NICHOLAS D. DURDIN, professor of metallurgy in the Polytechnic Institute of St. Petersburg. Prof. Durdin is a specialist in iron and steel, having been in charge of the open-hearth department of the Aboukoff Iron Works before being elected to his professorship, and having studied the metallurgy of iron under such lights as Chernoff, Heyn and Ledebur. While visiting America the professor has taken special work in metallography under Prof. Howe at Columbia, and is at present completing a course in the thermochemistry and metallurgy of iron with Prof. Richards at Lehigh University. Prof. Durdin says that the fundamental instruction given by some of our American technical schools in thermometallurgy and metallurgical calculations is the best in the world, and will be of ever increasing assistance in keeping American metallurgists at the head of their profession.

Digest of U. S. Patents.

Compiled by Byrnes & Townsend, Patent Lawyers, National Union Building, Washington, D. C.

ELECTRIC SMELTING AND REDUCTION PROCESSES.

(Continued.)

Nos. 609,466 and 609,467, Aug. 23, 1898, Auguste J. Rossi, New York City.

Produces a ferro-titanium containing more than 5 per cent of titanium, *e. g.*: (1) Mixes an ilmenite ore containing from 35 to 60 per cent of iron and 10 to 40 per cent titanic acid, especially ilmenorutile, 16 parts, with charcoal, 5 parts, and cast iron, 100 parts, and smelts the mixture at a temperature not less than 3500° F., producing an alloy containing iron 90, titanium 8, and carbon 2; (2) smelts a mixture of iron ores containing 20 per cent of titanic acid and 60 per cent of iron, 100 parts, with charcoal, 25 parts, producing an alloy of iron 77.4, titanium 16.5 and silicon 6.10; (3) smelts a mixture of the titaniferous slag produced by the process of his patent 486,941, 130 parts, with powdered carbon, 25 parts, and pig iron, 100 parts, producing an alloy containing titanium 27.53, silicon 2.30, iron 59.15, carbon 10.41; (4) smelts a mixture of the slag, 125 parts, with carbon and titaniferous iron ore, 160 parts. In all cases the charge must afford sufficient iron to maintain a bath of molten iron. The smelting may be effected by an oxyhydrogen flame or in an open hearth furnace. An electric furnace, however, is preferably employed, consisting of magnesia brick. The charge is placed in a graphite crucible, within the furnace chamber, the intermediate space being filled with pieces of charcoal. Adjustable electrodes pass through the side walls into proximity to the crucible. In operation, arcs pass between the electrodes to the crucible and between the pieces of charcoal. After the smelting is effected, the crucible is lifted out and the product is poured into a mold. From 50 to 100 pounds of alloy containing 15 per cent of titanium have been produced from one charge.

No. 624,041, May 2, 1899, Charles B. Jacobs, East Orange, N. J.

Produces soluble barium compounds by mixing barytes with sufficient carbon to reduce a part only of the barium sulphate to sulphide. When the mixture is electrically smelted, a barium sulphide is first produced and then reacts on the remaining sulphate, with the production of barium oxide and sulphur dioxide. The second reaction is incomplete, the product from a charge of barytes 20 parts, and carbon 1 part, containing barium oxide, 60 per cent, and barium sulphide 40 per cent. The production of a ton of the product requires 1636 kw-hours.

(To be concluded.)

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